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INFORMATION CIRCULAR

MILLING METHODS AND COSTS AT THE NACOZARI CONCENTRATOR OF THE PHELPS DODGE CORPORATION, NACOZARI, SONORA, MEXICO

BY

E. H. ROSE, AND W. B. CRAMER

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By E. H. Rose[2] and W. B. Cramer[3]

INTRODUCTION

This paper, describing the concentrator practice of the Moctezuma Copper Co., is one of a series being prepared by the Bureau of Mines.

The Nacozari concentrator of the Moctezuma Copper Co. (the Mexican subsidiary of the Phelps Dodge Corporation) is located about 75 miles south of Douglas, Ariz., in the Moctezuma District of the State of Sonora. Nacozari is the southern terminus of the Nacozari Railroad, over which all concentrate is transported to Douglas for smelting.

All of the tonnage treated by the concentrator is drawn from the company's Pilares mine, about 8 kilometers southeast of the mill, and connected therewith by a 36-inch gage steam railway.

The choice of mining method employed at Pilares depends upon the size and shape of the individual ore bodies, the grade of the ore and inclusions of waste in the ore, and the character of the wall rock. The ore generally supports itself well, but owing to the necessity for selective mining, weak waste walls, and irregularities in the size and shape of ore bodies, the horizontal cut-and-fill method predominates, though when the ore bodies are amenable, the shrinkage method is used. The classification by stoping methods throughout 1929 was as follows:

	Per cent
Horizontal cut and fill.....:	59.4
Inclined cut and fill (rill):	17.6
Square set..................:	17.7
Top slice...................:	3.7
Shrinkage...................:	1.6
Total ....:	100.0

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6358."

2 - One of the consulting engineers, U. S. Bureau of Mines, and mill superintendent of the Moctezuma Copper Co.

3 - One of the consulting engineers, U. S. Bureau of Mines, and consulting engineer, concentrating department, Phelps Dodge Corporation.

The present three-unit concentrator, of all steel and concrete construction, was designed and built by Mr. H. Kenyon Burch in 1921-1922, to have a capacity of 3,000 tons per day in three-stage concentration of roughing tables, flotation, and finishing tables. This was simplified to all-flotation in 1923-1924, by shutting down the tables, without much ultimate sacrifice in capacity. However, present mill feed from lower mine levels is materially harder and more difficult to grind, which fact has dropped the economically most advantageous milling rate to 2,700 dry tons per 24 hours of operation. The construction of 1921 was considered to be "remodelling" by virtue of being upon the site of the older mill which had been operating up until that time, and which had also been built by Mr. Burch, in 1908. However, the older mill was almost completely dismantled, and little more than the foundations and the building were utilized in the remodelling. Extensive pilot plant operation in 1919 led to the development of the three-stage flow sheet, as adopted in new construction and as used in the initial operations in 1922. Acting in cooperation with Mr. F. E. Marcy, an important phase of pilot plant operations was the development of rod mills of a larger size than had been considered practicable before that time. These were subsequently adopted for general use not only at Nacozari, but also at the Corporation plants at Bisbee, Ariz., and Tyrone, N. M.

Mill water, in the quantity of about 3,000,000 gallons per day, is made up of about 45 per cent of new water, 20 per cent of water reclaimed from the concentrates dewatering circuit, and 35 per cent reclaimed from the tailings pond. The new water comes from the Huacal dam and reservoir, 3 miles northeast of Nacozari, where a concrete, single-arch dam, approximately 100 feet high, gives an impoundage capacity of 1,000,000,000 gallons of flood waters. The spillways have overflowed practically every summer rainy season since the dam was constructed in 1911. Of the total annual precipitation, amounting to 20 to 25 inches, about half falls during July and August.

Power is obtained from a Diesel engine plant at Nacozari. This consists of four 1,250-horsepower and two 22,00-horsepower Carols-type, two-stroke cycle engines, giving a total rated capacity of 9,400 horsepower. Power is transmitted to the mill at 6,600 volts and is there stepped down to 2,200 volts for all Marcy mill motors and to 220 volts for all other motors.

ORE TREATED

The gangue of the ore above the 700-foot level of the mine, now constituting about 20 per cent of the total output, is a quite hard latite breccia, and that of the remaining 80 per cent from the lower levels is an extremely hard andesite breccia, considered to be as hard as any copper ore in the Southwest. The latite and andesite flows have been cut by intrusions of monzonite, diorite, and quartz porphyry. The economic mineral is chalcopyrite, accompanied by minute quantities of chalcocite, covellite, and bornite. About 0.05 per cent of acid-soluble copper is present, chiefly tenorite, with some cuprite. The ratio of pyrite to chalcopyrite in the ore is now about 2 to 1. Some spalerite, magetite, and hematite are also present. Silver, associated almost exclusively with the chalcopyrite, runs about three-fourths of an ounce per ton. Some of the alumina in the mill heads appears to be reactive, and affects slime metallurgy chemically and as colloid. Following is a chemical analysis of mill heads, with the combinations of the more significant minerals, as calculated from parallel chemical and miscroscopic analyses:

Chemical analysis of mill heads

	per cent		per cent
Cu (total)	2.63	Chalcopyrite	7.22
Cu (acid-soluble)	0.05	Chalcocite and bornite	0.10
Fe (total)	11.54	Tenorite, etc	0.10
S	8.70	Pyrite	11.53
SiO_2	48.58	Nonsulphide gangue	81.05
Al_2O_3	13.70		
CaO	1.88		
MgO	2.25		
Ignition loss and unaccounted for	10.67		
Total	100.00	Total	100.00

A screen analysis of the product from the crushing plant to the primary grinding mills is shown as Table 3. The coarseness of this feed does not signify a belief that it would not be economical to crush to a finer size preceding the grinding mills. It is possible that changes will ultimately be made whereby this product will be reduced to one-half inch or thereabouts.

Aside from the output of a few of the upper mine levels during the rainy season, the ore received at the mill is so dry as to cause no difficulty in choking of any of the crushers. Indeed, its dryness gives rise to a considerable amount of dust in the dry crushing operation, and has necessitated the installation of a complete dust-collecting system there, with ducts extending to the several sources of dust, which are housed in and exhausted by means of a suction fan. About 95 per cent of the dust drawn out by the fan is recovered in a spray tank and is sent into the slime flotation circuit.

The moisture content of the ore has averaged about 3.25 per cent for several years. This relatively low moisture content is, of course, advantageous metallurgically, in the sense that it gives rise to less oxidation and chemical change on the part of broken ore in stopes. In 1927 the mine drainage system and the ventilation system were both extended and improved, to the end that no water now comes in contact with the ore in dumps or ore passes. While this change was primarily for the sake of bettering mining conditions, the cost of such improvements in the mine was probably more than repaid by their favorable influence upon metallurgy of the mill, as oxidation and generation of detrimental salts in the ore have been less. Among the factors that effect these changes in the ore are the time allowed after breaking, the moisture present in the ground and the air, the mineral surface exposed, and the amount of iron present as a generator of solvents for the copper sulphides. In this respect, at least, "hard ore" is actually an advantage, for a hard ore is usually a less osidized ore.

The chemical behavior of ore in the mine, particularly of the fines, forms an integral part of the general picture of flotation when the latter process is viewed from the chemical standpoint. At Nacozari, the belief has for some time been held that the governing features of flotation are predominantly chemical phenomena, even though the process itself be not strictly chemical in its entirety.

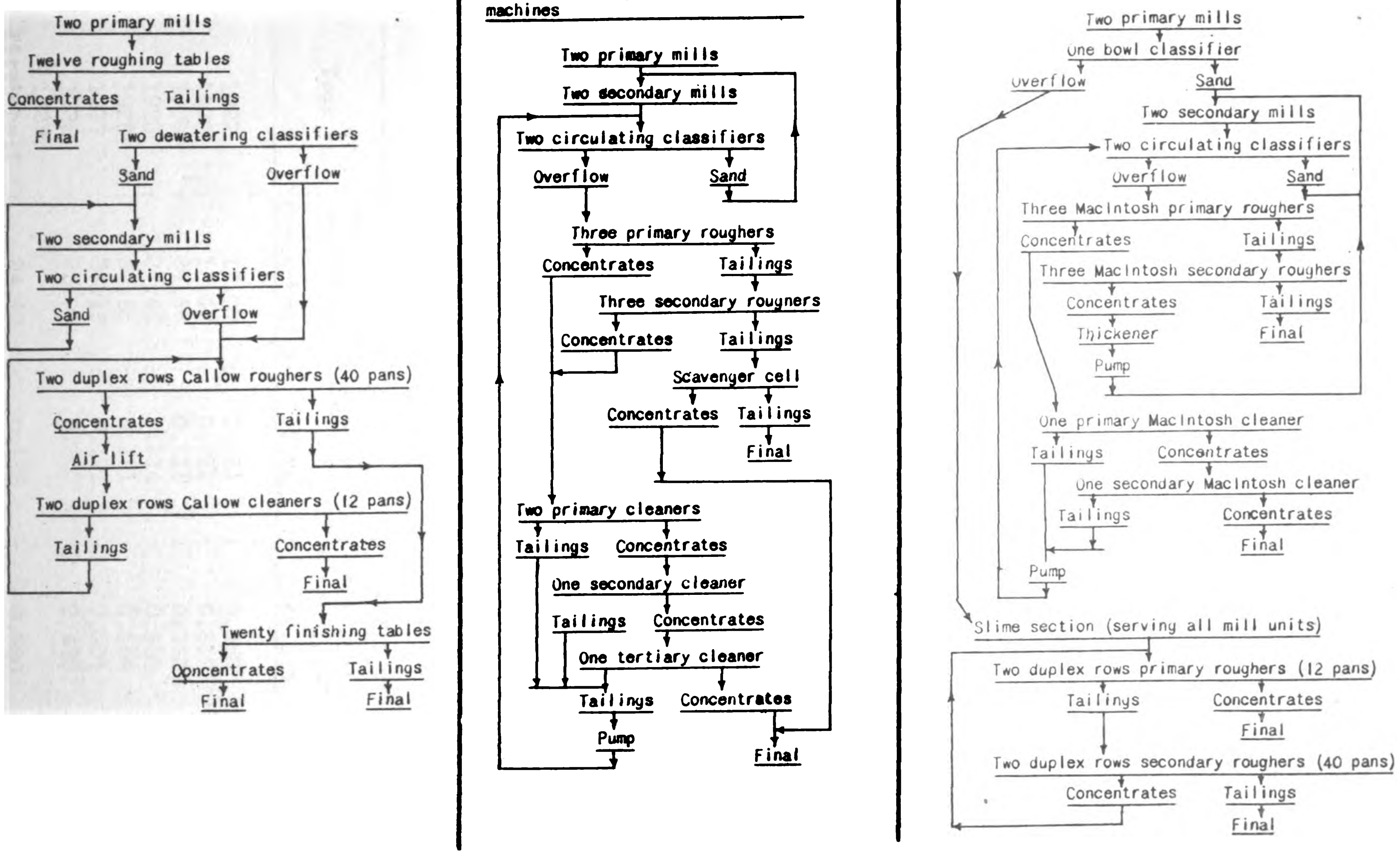
Given sufficient liberation of significant minerals by grinding, local evidence confirms the almost universal finding of mill experience that, with the important exception of middlings, the most difficult portion of the ore to treat is the original mine fines, but by treating them in a separate circuit—a practice begun in July, 1929, coincident with the installation of bowl classifiers—better tailings are made upon this portion of the mill feed than upon any other. The feed to this separate slime section - comprising 30 to 33 per cent of the total mill feed tonnage - averages 0.11 per cent of "oxide copper" as compared with 0.06 per cent for the general mill heads, and on that section the recovery of oxide copper averages 60 per cent. The mine run of ore to the mill averages 7 per cent minus 100 mesh; that from the crushing plant to the primary mills, 10 to 12 per cent and that from the primary mills, 30 to 33 per cent, which is the feed to the separate slime section.

In laboratory flotation tests upon ore from one or two stopes which happen to be especially oxidized, a water-wash prior to milling has been found very advantageous in reducing lime consumption and improving recovery, though this process is still in an experimental stage.

At the smelter at Douglas, the Corporation maintains a centralized and fully equipped microscopic laboratory in charge of a trained microscopist. To this laboratory, samples of mill products are sent from the several branches at stated intervals, for combined microscopic and chemical analyses. This laboratory has been functioning regularly for about two years. The reports and information obtained have been found very useful in relative comparisons. In the past, the examinations have been made on polished surfaces of grains embedded in a matrix; they deal with plane sections of grains, rather than the whole-grain entities which are being dealt with actually in the concentrating process.

Until recently, tailings have been the usual product sent from Nacozari, to aid in determining the nature and cause of tailings losses. Concentrates and flotation middlings have also been so studied. A microscopic examination of mill feed is also in process, but it is not available at the date of writing. The concentrator organization has an excellent microscope by means of which more superficial but more frequent studies are made.

No results of microscopic studies are given in this present paper, however, because it is felt that this work, or at least the interpretation of it, is still in a development stage. It has been found that magnifications of the order of 1,200 diameters reveal many inclusions which are totally invisible at 125 diameters. Thus, while the smallness of the field at the higher magnification may introduce a "sampling" error, the more representative field of the lower magnification does not always yield accurate metallurgical information.



ORIGINAL FLOW SHEET - 1922

Showing one of three identical mill sections, with 3-stage concentration of roughing tables, flotation, and finishing tables

FLOW SHEET AT CLOSE OF 1926

Showing one of three identical mill sections, after elimination of all tables and development of all-flotation flow sheet, with triple cleaning of concentrates and all Callow cells replaced by MacIntosh machines

FLOW SHEET AS OF MARCH 1, 1930

Showing one of three identical mill sections and general slime section incorporating bowl classifiers, separate treatment of primary slimes, and experimental regrinding of the MacIntosh secondary rougher concentrates

Figure 1.- Panels showing evolution of flow sheet of the Nacozari concentrator, from 1922 to 1930

HISTORY OF CONCENTRATOR OPERATIONS

Metallurgical progress at Nacozari has come about through a series of quite definite, successive steps, some of which are diagrammed in the panel flow sheets of Figure 1 and may be enumerated as follows, in comparison with concurrent milling data:

Change

1. Lime circuit adopted, 1922.
2. Finishing tables shut down, 1923.
3. Double cleaning of concentrates adopted, 1923.
4. Roughing table concentrates experimentally reground and selectively floated, 1923. This step never permanently incorporated into flowsheet.
5. Roughing tables shut down and all-flotation flow sheet established, 1923.
6. Triple cleaning of concentrates adopted, 1924.
7. Change made to xanthate reagents, 1925.
8. All Callow cells replaced by MacIntosh cells, 1926.
9. Bowl classifiers installed, June, 1929.
10. Separate treatment of primary slimes begun, July, 1929.
11. Tonnage rate per hour reduced to maintain grinding on harder ore, 1929.
12. Nitrogen compounds studied as mill flotation reagents, 1929.
13. Thickening, regrinding, and separate retreatment of flotation middlings experimentally begun, February, 1930.

The affect of some of these changes on the metallurgy as expressed in copper content of concentrates and tailings and also on costs is shown in the following tabulation:

Year	Analyses Per cent copper Concentrates	 Tailings	Direct mill cost, dollars per ton
1922	12.07	0.221	0.952
1923	18.42	0.271	0.818
1924	27.68	0.218	0.763
1925	27.46	0.214	0.600
1926	27.86	0.191	0.564
1927	27.18	0.171	0.504
1928	25.65	0.207	0.470
1929:			
First half	24.47	0.198	0.520
Second half	23.08	0.126	0.590

Other steps which are in mind for possible future investigation include finer crushing; a partial closed circuit of primary mills; insertion of a stage of flotation between secondary mills and secondary mill closed circuit classifiers, and the production of a high-density flotation froth for direct filtration without intermediate thickening. The last-named step is actually now in effect upon two mill sections, as will be noted under the subject of concentrates handling.

Although those who were then in charge of operations were pioneering in most of the earlier steps mentioned, the modes of development during the years 1922, 1923, and 1924 are as typical of what has now become concordant history in most of the major copper concentrators of the Southwest as to require little elaboration in the present paper.

It is noteworthy that the Nacozari concentrator is believed to be the first copper concentrator in the Southwest - not to speak more generally - to apply selective flotation commercially to its type of ore. Due largely to the smelter's preference for coarse concentrates, the design and construction of the mill in 1920-1921 did not propose any marked selectivity in the recovery of copper in preference to iron. It was predicated that flotation would be in natural circuit - that is, neither acid nor alkali would be added. When the operations began, however, experience soon showed that the use of lime was dictated to benefit recovery; and we now know that lime was needed to stop the generation of solutions of iron salts, thus decreasing the tendency of those salts to react from solution to consume the collecter reagent, rather than from the mineral surface to produce difficultly soluble cuprous films. Lime was also very useful in decreasing consumption of grinding rods and liners, which in the natural circuit without the chemical protection afforded by lime were rapidly eaten out by electrolysis and required distressingly frequent replacement. It was not until a year later, however, after all tables had been taken out of service, that lime began to play its present important part in selective flotation.

The shutting down of all finishing tables following flotation was the first definite step toward the present all-flotation flow sheet. As these tables treated flotation tailings directly, their field for useful effort was rapidly restricted as flotation recovery improved. After six months of operation, they were shut down in January, 1923, by which time the flotation tailings had dropped to 0.25 per cent of copper. They were making pyritic concentrates so low in copper that they would not pay their way to the smelter and which, as flotation continued to improve, soon would not have contained enough copper to warrant regrinding or other forms of re-treatment, especially in view of the great amount of pyrite which was only awaiting the opportunity to appear as table concentrates. In the light of later events, it has been the selective grinding accomplished by the action of Dorr classifiers upon pyritic feed which permanently removed the necessity of tabling out pyrite for further treatment.

When double cleaning of concentrates was introduced in April 1924, selective flotation at Nacozari began to show its real value. Prior to that time, the singly-cleaned froth averaged about 18 per cent copper, and was carrying a considerable percentage of copper-free waste, which merely had not had the opportunity to drop out, in the hurried environment of the roughing operation. An extra stage of cleaning immediately brought the grade to 24 per cent copper without increasing the grade of tailings. A year or so later, a third stage of cleaning was introduced, which graded the concentrates up to 27 per cent of copper, and which similarly did not increase the tailings assay (though possibly delaying the day when copper content in the tailings could be further reduced).

It was always apparent that the average grade of final mill concentrates could never be brought up to anything even approaching the theoretically possible, so long as half the total mill concentrates was coming from roughing tables which simultaneously and indiscriminatingly recovered nearly half of the pyrite in their feed with as much of the chalcopyrite. Two remedies were possible: to continue the use of roughing tables, but to regrind and then selectively float their concentrates, or to shut them down completely and throw their entire burden directly upon flotation, incidentally increasing the feed to the secondary mills (and the work required of these mills) by the tonnage which until then had been roughing table concentrates.

Had there been extensive association between chalcopyrite and pyrite in the mill feed, and little between chalcopyrite and silica, the first method would probably have been dictated, for it would have given - in effect - a selectively finer grinding of the chalcopyrite and pyrite.. On the other hand, had the reverse been true with chalcopyrite-silica liberation being the dominant grinding problem, the making of pyritic roughing table concentrates for grinding would have been pointless.

As it has turned out, the retention of roughing tables for the purpose above intimated would have been unsound practice, despite the fact that it is chalcopyrite-pyrite association which increases from year to year as deeper mine levels are reached. Most of the pyrite and most of the chalcopyrite break free from each other in the size range in which the tables would have been working, and grinding effort upon those freed portions would have been uselessly expended. Thus while chalcopyrite-pyrite association does indeed persist at the present fineness of grinding, it is confined to a relatively small bulk of mineral which can be, and is, segregated by flotation methods as middlings for regrinding. Tables could not have been made to draw so fine a distinction at an acceptable cost.

Of course, complete liberation will probably always be economically impossible when - as stated on page 12 - there is some association so fine that it does not even become visible at less than 1,000 to 1,200 diameters magnification.

For a period of two or three weeks in 1923, the scheme was practiced of a fine regrinding of roughing table concentrates and a coarse regrinding of roughing table tailings, mixing the two products for flotation treatment. Very satisfactory and economical results were obtained, but various conditions of mechanical layout argued the mere shutting down and by-passing of the roughing tables as preferable. That was done and continues to the present time, and no reason to regret the decision has ever made itself apparent.

Thus the all-flotation flow sheet was brought about without the addition of any new equipment. For a time, general mill recovery suffered and unit capacity was cut, but in the final measure of net profit per ton of ore, the method far more than justified itself by virtue of the decreased concentrates tonnage upon which freight and smelting charges had to be paid. Parenthetically, it may be stated here that ultimate financial gain - the net profit resulting from a ton of ore at Nacozari - has always been the guide for procedure. Endeavor is made to put together that combination of operating costs, concentrates grade, and tailings assay which will yield the highest profit, even though at any time such a course may require sacrifice in any one of the three to produce a greater commensurate gain in any of the other two. The local importance of this is accentuated by high realization costs.

From the smelter viewpoint, the obvious advantages of high-grade concentrates, coupled with fortunate developments in smelting practice, soon eased the demand for a coarse mill product. Looking backward, it is apparent that the application of selectivity was profitable from the very outset. Once established metallurgically, it was a vital principle which could not be abandoned. Nevertheless, a very serious mechanical difficulty was immediately encountered. The flat-bottomed Callow-Inspiration cells were not suitable for selective work on relatively coarse feed. The rejected pyrite settled immovably to the bottom, cut off the air supply, and caused a serious handicap to copper recovery through sheer aeration impedence. This seems to have been the universal experience wherever Callow-Inspiration circuits were made selective.

There was but one remedy: a new type of cell must be developed. The MacIntosh cell was the result. This machine, originally conceived by John C. MacIntosh, then general mill foreman, and developed and improved by Wm. T. MacDonald, then mill superintendent, retained the porous mat of the Callow cell, but set it in motion as a porous, revolving rotor, gently stirring the pulp by mechanical means and thus keeping the pyrite in suspension and in motion through the cell. Thirty of the new machines were built locally and the entire mill tonnage began passing through them instead of the Callows early in 1926. Two of the three units of Callows were torn out to make room for this change; the third Callow unit was left idle, but was not removed.

Coal tar and pine oil were the flotation reagents used during the first two years of operation. Late in 1924, thiocarbanalid in orthotoluidin replaced the coal tar and continued at the entire mill as the collector reagent for about a year. Following experience elsewhere, a change to xanthate was made in 1925, and the sodium ethyl salt is the present standard reagent, except as some mill sections may be given over to other experimental reagent work.

In extensive mill tests, the thiophosphates were found to give very satisfactory recovery, contingent upon a closely controlled metallurgical environment. They, in common with the xanthates and other sulphur reagents, possess the shortcoming of reacting with pyrite when slight excess of reagent is present; this behavior does not appear to be shown by the nitrogen reagents.

When the roughing tables were shut down, the two stages of grinding with which the mill was equipped then became immediately contiguous in the flow sheet, and the total mill feed necessarily had to go from the first to the second without intermediate classification to remove finished material. Until later years, when harder ore began to exact its toll from grinding capacity, it was satisfactory - even if not technically correct - so to operate. In 1928, however, the need for an intermediate stage of classification began to be felt acutely, and a new 15-foot bowl classifier to remove minus 65-mesh material was experimentally inserted into the flow sheet of one mill unit between the two stages of grinding. The bowl overflow then rejoined the sands after the latter were ground in the secondary mills, so that the function of the bowl at first was merely to by-pass finished material around the secondary grinding circuit. In this manner of operating, the bowl showed an improvement in grinding efficiency equivalent to a saving of about 5 cents per ton of ore, and consequently two more bowl classifiers were ordered, correspondingly to equip the remaining two units. The three mill units have been operating with bowl classifiers since June 1929.

Before these bowls began operating, it was realized that their overflow would be of a different chemical character from their sands, even after the latter had been reground, for the bowl overflow contains practically all of the original oxidized fines of the mine ore, in addition to the fines produced by the primary mills. It was thought that some advantage might be gained by sending the bowl overflow to a separage flotation circuit. That was begun experimentally in April, 1929, using the one bowl classifier which was then in service and utilizing part of the old Callow flotation unit which had been idle since 1926, and which was thereupon reconditioned for the test. Such an improvement in over-all mill recovery was immediately apparent, that the method was extended to all three mill units as soon as the bowl classifiers were installed, and is now a permanent part of the regular mill flowsheet. The overflow of the three bowl classifiers join and is treated in the Callow unit, while all of the reground sands continue to be treated in the MacIntosh units. The merits of this procedure are discussed at a later point in this paper.

It has been found advantageous in many flotation mills to pass all mill tonnage through bowl classifiers, which exert a peculiarly uniform control upon the size of flotation feed. This was recently tried at Nacozari, but did not result favorably, as such a flow sheet necessarily abandoned separate slime treatment, for under those conditions, the bowl overflow constituted the entire flotation feed. Loss of the advantages of separate slime treatment more than offset the advantage of greater uniformity of grinding, so that while grinding upon the same tonnage was actually somewhat finer, the recovery was nevertheless lower than under the normal separate-slime-treatment flow sheet. Taking all flotation feed from bowl classifiers would probably be advantageous at Nacozari if separate primary slime treatment could at the same time be retained, but as that would require additional classifiers, the idea is not now being entertained.

Even with the bowl classifiers, grinding capacity has not been restored to its old figure because of increasing hardness of ore, and operating costs have accordingly been higher in 1929. This is not surprising, since the mill feed from the crushing plant contains material as coarse as 1-3/4 inch, and the proper mode for future attack would seem to be either provision of finer crushing, or of regrinding of flotation middlings. Nacozari experience indicated that 1-3/4 inch is not the economic size of crushing ahead of rod mills when the ore is hard.

PRESENT METHOD OF CONCENTRATION

Figure 2 shows the general flow sheet of the concentrator and of the coarse crushing plant, and Figure 3 shows a general cross-section of the concentrator.

Flow Sheet of Any One Section

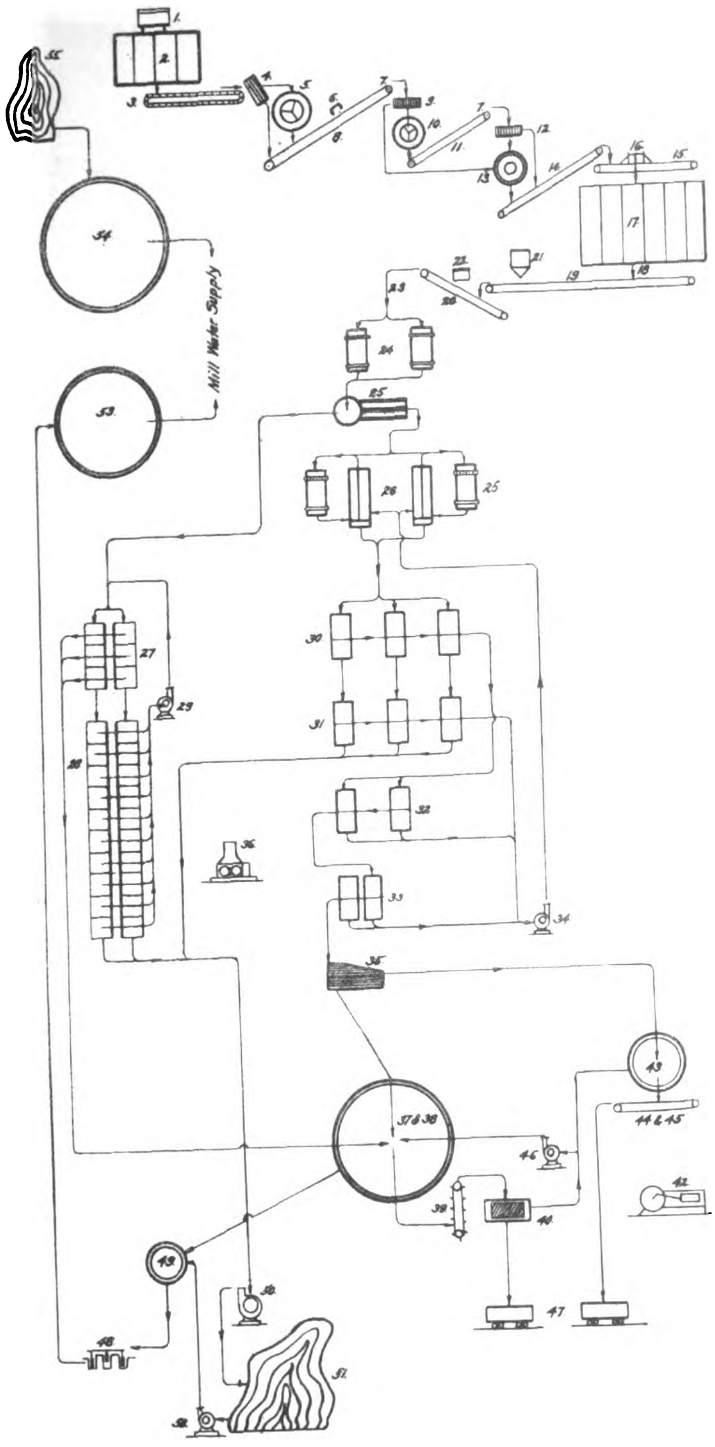
The Nacozari concentrator has three general sections, in addition to the separate slime flotation section. The three are identical and each has a capacity, under present conditions, of 880 dry tons per 24 hours, when grinding to about 13 per cent on 65-mesh. The ore is drawn from the 8,000-ton capacity fine ore bin by 34 apron feeders and is conveyed into a 20-ton capacity surge bin. This bin contains three compartments, each completely partitioned off from the other two. The entire ore stream may be turned in quick succession into any one of the three compartments, so that the allocation of ore to the three mill units is a time division rather than a division of stream of ore. The three compartments of the surge bin discharge by 20-inch apron feeder (one in the bottom of each compartment) upon unit feed conveyors leading to the respective mill units. Each of the unit feed conveyors passes over a Merrick Model-E weightometer, whose readings, taken at the end of every shift, indicate tonnage treated by each unit. The drawing of ore from the fine ore bin is controlled from the primary mill floor by four push-buttons, each of which starts or stops a bank of feeders.

Each unit feed conveyor discharges into a splitter box, where the unit tonnage is divided into two halves, each half going to one primary 6 by 12 foot Marcy roller mill, of which there are two per unit, or six for the entire mill.

The primary mills work in single pass and the discharge of each pair joins and goes into one 13-foot Dorr bowl classifier, of which there is one for each unit. The overflows of the three bowl classifiers join and go into a trunk launder to the general slime unit (later discussed) which, except for the source of its feed, is independent of the three units now being described.

The sand discharge of each bowl classifier is laundered to, and divided between two secondary 6 by 12 foot Marcy rod mills, or six for the entire mill. These mills are each in closed circuit with a 6 by 23 foot, 4-inch Dorr duplex classifier. The classifiers overflow at about 3 per cent on 35 mesh, and the joined overflow of each pair of classifiers is laundered to a MacIntosh flotation unit. Dilution water in the classifiers is made up chiefly of returned flotation middlings.

Each of the three sand flotation sections is made up of three primary and three secondary roughers, and of two primary, one secondary, and one tertiary cleaner, a total of 10 MacIntosh machines per unit. The unit flotation feed is divided evenly to the three primary roughers, whose tailings go into a corresponding secondary rougher. The tailings of the latter are final and go to the tailings pond for final disposition and storage. The primary rougher concentrate is cleaned in three successive stages in the primary, secondary,



No.	INDEX
1	25-ton capacity railroad cars from mine
2	Coarse ore bins, 1,800 tons capacity
3	Four Webster pan feeders, 48"×14'6"
4	Four bar grizzlies, 4-inch spacing
5	Two No. 8 McCully gyratory crushers
6	Two suspended magnets, 250 volts D.C.
7	Two Dings magnetic head pulleys, 250 volts D.C.
8	Conveyors Nos. 1 and 3, 36"×53', 260 f.p.m., 6 ply
9	Four fixed ring grizzlies set at 1 3/4" and 5/8"
10	Two No. 4-F Telsmith crushers set at 1 1/2"
11	Conveyors Nos. 2 and 4, 36"×22', 200 f.p.m., 6 ply
12	Two Dings magnetic head pulleys
13	Two vertical shaft Symons disk crushers
14	Conveyor No. 5, 24"×350', 20° slope, 6 ply
15	Conveyor No. 6, 24"×180', horizontal
16	S-A automatic tripper
17	Fine ore bins, 8,000-tons capacity
18	34 Webster pan feeders, 20'×2'4"
19	Conveyors Nos. 7 and 8, 18"×180', 208 f.p.m.
20	Conveyors Nos. 9 and 10, 20"×52' abd 51'
21	Lime feeder (dry)
22	Three Merrick weightometers
23	Conveyors Nos. 11, 12, and 13; 115', 48', 48'
24	12 Marcy roller mills, 6'×12', 17.5 r.p.m.
25	Three 13-foot Dorr bowl classifiers
26	Six 6'×23'4" Dorr circulating classifiers
27	12 Callow-type flotation pans, 46"×46"
28	40 Callow-type flotation pans, 46"×46"
29	Two 4-inch middlings return pumps
30	Nine 10-foot Macintosh flotation cells
31	Nine 10-foot Macintosh flotation cells
32	Six 10-foot Macintosh flotation cells
33	Six 10-foot Macintosh flotation cells
34	Three return middlings pumps
35	Three finishing tables for concentrates distribution
36	One No. 5 1/2 and one No. 7 Root blowers, flotation air
37	One Dorr thickener, 50-foot diameter; area, 3,300 sq. ft.
38	One Dorr thickener, 60-foot diameter; area, 4,241 sq. ft.
39	One bucket elevator, 18"×115'
40	Two 14'×14' Oliver continuous filters
41	One 14"×12" No. 12 air compressor
42	One I-R type XRB vacuum pump
43	Eight concrete concentrates storage bins, 122 tons each
44	Conveyor No. 14, 24"×230', 350 f.p.m.
45	Conveyor No. 15, 24"×80', 350 f.p.m., 30° slope
46	Three centrifugal pumps, two 4" one 2 1/2"
47	80 F.C. de N. 55-ton standard gondolas
48	Two 14"×12" Aldrich triplex pumps
49	Six concrete storage tanks
50	Four 4-inch centrifugal pumps for tailings elevation
51	Tailings and water storage dam
52	One 4-inch centrifugal return water pump
53	Three return water storage tanks
54	Mill fresh water supply tank, 500,000 gallons
55	Huacal dam storage capacity, 900,000,000 gallons

Belt lengths as given are between centers

Figure 2.- Flow sheet and active equipment list for the last six months of 1929, Nacozari concentrator

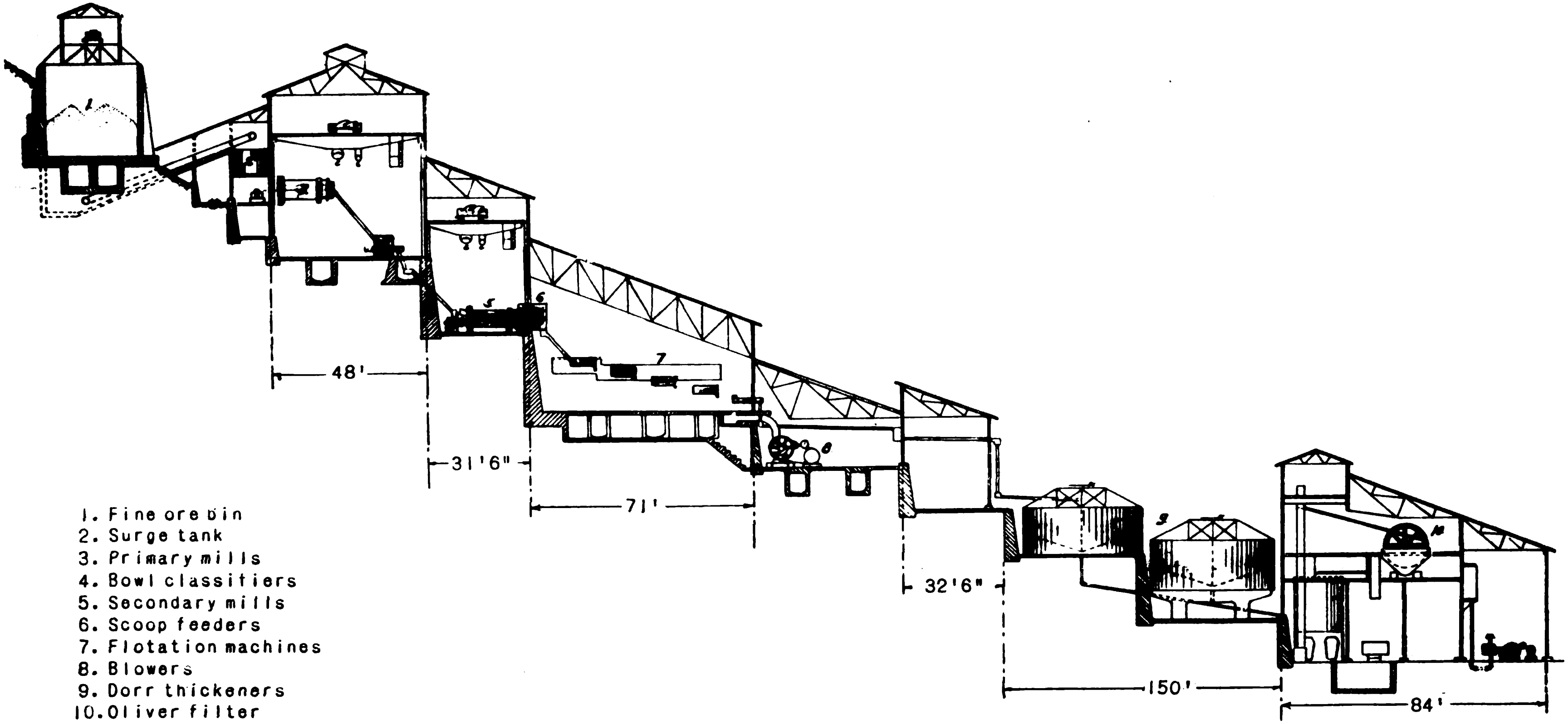


Figure 3.- General cross section of Nacozari concentrator

and tertiary cleaners. The tailings of all of these cleaners, together with the concentrates of the secondary roughers, go to a common sump and are pumped to the circulating classifiers on the secondary mill floor, where they serve as dilution water and then repass through the flotation plant. However, it is not considered to be correct in principle merely to recirculate flotation middlings without some interposed change of its condition, and to that end the regrinding of this product is now being practiced experimentally, with very gratifying results. The MacIntosh cells have been modified to produce a thicker middlings product which is returned directly into the secondary mills. Regrinding of the circulating middlings, however, falls short of eliminating the middling problem, for the disparity between the grade of concentrates finally produced and that theoretically possible is not due to dilution by either free waste or by a serious quantity of the middlings which has circulated, but rather to material as truly middlings as that which circulates but which the cleaners do not reject because the individual grains contain sufficient chalcopyrite for them to float through the circuit to concentrate.

Since July, 1929, the joined overflow of the three bowl classifiers, at 2 per cent on 65 mesh, has been going to the slime section. This section, which has no grinding or classifying equipment, consists of two duplex rows of Callow-Inspiration cells, the four single rows each containing thirteen 42 by 42 inch pans, covered with 3-ply canvas mats. The concentrates of the first three cells of each row are the final slime section concentrates. The tailings of these cells go directly on through the remaining ten cells, from which they emerge as the final slime section tailings, and then join the sand section tailings for laundering to the tailings pond. The concentrates of the final 10 cells is pumped back to the head of the slime circuit; this product also may ultimately receive other treatment than mere recirculation.

Until recently it was the practice to table the flotation concentrates, taking out a coarse product which could be dewatered by gravity. Recently this practice has been abandoned, as it is found that the general average moisture can be held about 2 per cent lower by filtering the entire tonnage. As the final MacIntosh section cleaners produce a froth which now approximates 50 per cent of solids, the concentrates of two mill units is going directly to filters without thickening.

The respective divisions of the concentrating process are now detailed.

Breaking

Ore is delivered from the mine to the mill by narrow-gage steam railway, and is hauled in trains of 12 cars, each car holding about 32 tons of ore. As the mine grizzlies are now set at not to exceed 16-inch openings, that is the maximum size of rock received at the mill. After dumping from the mine railroad cars into the 1,800-ton-capacity coarse ore bin which is part of the crushing plant, the ore is drawn out by four 48-inch by 14-foot, 6-inch pan feeders.

The present crushing plant handles a total of 190 tons per hour, and is divided into two identical sections, with two of the pan feeders serving each section. Only one pan feeder operates at a time on each section. The pan feeders discharge upon a fixed bar grizzly, 45° slope, with a 4-inch spacing between bars. The following paragraphs describe either one of the two crusher sections.

The grizzly oversize, consisting of 70 per cent of the total feed, drops into the hopper of a No. 8 McCully gyratory crusher, set at 4-inch opening. Each of the two McCully crushers is belt-driven from a 75-horsepower motor and makes 148 gyrations per minute.

On each crusher section, the bar grizzly undersize joins the McCully discharge upon a 36-inch conveyor which has a 20° upward slope. This belt passes beneath a suspended electromagnet, which removes a considerable portion of the tramp iron; the belt also discharges over a Dings magnetic pulley. A motor-generator set just outside the crusher building supplies direct current for all electromagnets.

Intermediate Crushing

The above conveyor discharge drops over a fixed ring grizzly, with 3/4-inch rings spaced 1-3/4 inch apart. The grizzly oversize, consisting of about 15 per cent of the original ore, drops into a No. 4-F Telsmith reduction crusher. This machine, designed for a coarser size range than that in which it works, accomplishes very little reduction. It is belt-driven by a 40-horsepower motor, and makes 110 gyrations per minute.

The undersize of the 1-3/4-inch grizzly drops upon a second ring grizzly whose spacing is 5/8 inch. The undersize of the latter goes by gravity chute to the general conveyor leading to the fine ore bins. The oversize joins the Telsmith product upon a short inclined 36-inch conveyor which discharges over a magnetic pulley, upon a third ring grizzly, whose spacing is also 5/8 inch. The fines go directly upon the final product conveyor to the fine ore bins, while the oversize drops into a Symons 48-inch, vertical shaft, disk crusher for the third and final stage of crushing. These Symons crushers are of an old type built by Chalmers and Williams in 1916.

The size of the crushing plant product is shown below as the primary mill feed, which, of course, is the same product.

Primary Grinding

The two 6 by 12 foot Marcy primary rod mills on each of the three mill sections operate as a pair, one rotating as a right-hand mill and the other as a left-hand mill, though interchangeable and having identical liners. All mills are driven by 2,200-volt induction motors through Falk double-reduction herringbone gears. The mills are driven at 17-1/2 revolutions per minute by spur gear and pinion from the slow-speed shaft.

Feed is received through a gravity spout into the trunnion of the mill, and is assisted into the mill by a trunnion screw. All of the primary mills have a 12-inch intake opening, and a 30-inch discharge opening.

With feed of the present size each mill handles 440 dry tons per 24 hours. The primary mills have always operated in open circuit, but a test is planned to determine the advantage of a partially closed circuit. The primary mills operate at 78 per cent of solids. A typical screen analysis of their feed and discharge is as follows:

Screen analysis of feed and discharge of primary mills

	Primary mill feed, Per cent weight retained		Primary mill discharge, Per cent weight retained	
	This mesh	Cumulative	This mesh	Cumulative
Plus 1-1/2 inch round..	4.42	4.42		
Plus 1-1/4 inch round..	5.71	10.13		
Plus 1 inch round......	8.64	18.77		
Plus 3/4 inch round....	10.19	28.96		
Plus 1/2 inch round....	11.79	40.75		
Plus 3 mesh Tyler......	11.03	51.78		
Plus 4 " "	7.07	58.85		
Plus 6 " "	6.95	65.80	0.35	0.35
Plus 8 " "	4.65	70.45	0.93	1.28
Plus 10 " "	4.20	74.65	2.77	4.05
Plus 14 " "	3.89	78.54	7.57	11.62
Plus 20 " "	2.85	81.39	8.67	20.29
Plus 28 " "	2.89	84.28	11.82	32.11
Plus 35 " "	1.85	86.13	9.05	41.16
Plus 48 " "	1.71	87.84	8.52	49.68
Plus 65 " "	1.38	89.22	5.28	54.96
Plus 100 " "	2.36	91.58	10.12	65.08
Plus 150 " "	0.97	92.55	4.77	69.85
Plus 200 " "	0.49	93.04	2.57	72.42
Minus 200 " "	6.96	100.00	27.58	100.00

The discharge of each pair of primary mills is deslimed by one 13-foot Dorr bowl classifier. With the feed shown in the above screen analysis, about 31 per cent, by weight, overflows as finished feed for the slime flotation section at 14 per cent of solids. Due to its higher assay, the overflow carries about 38 per cent of the mill input copper. The remainder, comprising the bowl sands, is laundered directly to the secondary mills. Screen analyses of the two bowl classifier products are as follows:

Screen analyses of products of the two bowl classifiers

Mesh	Bowl sands, per cent weight retained		Bowl overflow, per cent weight retained	
	This mesh	Cumulative	This mesh	Cumulative
Plus 6 mesh............	0.50	0.50		
Plus 8 "	1.18	1.68		
Plus 10 "	5.08	6.76		
Plus 14 "	10.48	17.24		
Plus 20 "	12.38	29.62		
Plus 28 "	16.70	46.32		
Plus 35 "	12.81	59.13		
Plus 48 "	11.87	71.00	0.73	0.73
Plus 65 "	8.50	79.50	1.23	1.96
Plus 100 "	12.50	92.00	4.37	6.33
Plus 150 "	4.00	96.00	4.50	10.83
Plus 200 "	1.38	97.38	3.17	14.00
Minus 200 "	2.62	100.00	86.00	100.00

The so-called secondary mills are identical with the primary mills as to design, size, drive, speed, and shell liners. Instead of having gravity feed chutes, however, each takes its feed through a single scoop, as a means of elevating the sands of the classifier with which it is in closed circuit. The mill discharge is also elevated into the classifier by means of a double-scoop elevator of 4-foot radius. While a 30-inch discharge opening has always been found advantageous on the primary mills, an 18-inch opening is preferred on the secondary mills. The secondary mills operate at about 73 per cent of solids and the circulating classifier overflow at 20 per cent solids. Following are typical screen analyses of the various products in the secondary grinding circuit, when working on bowl sands of the screen analysis already shown:

Screen analyses of various products in secondary grinding circuit

Mesh	Secondary mill discharge, per cent weight		Circulating classifier sands, per cent weight		Circulating classifier overflow[1] per cent weight	
	This mesh	Cumulative	This mesh	Cumulative	This mesh	Cumulative
Plus 20 mesh.......	0.30	0.30	0.70	0.70		
Plus 28 "	1.80	2.10	3.90	4.60		
Plus 35 "	5.60	7.70	8.10	12.70	0.40	0.40
Plus 48 "	12.30	20.00	18.80	31.50	5.30	5.70
Plus 65 "	12.30	32.30	17.10	48.60	6.90	12.60
Plus 100 "	26.10	58.40	31.40	80.00	16.90	29.50
Plus 150 "	11.20	69.60	11.70	91.70	8.40	37.90
Plus 200 "	4.20	73.80	2.90	94.60	4.10	42.00
Minus 200 "	26.20	100.00	5.40	100.00	58.00	100.00

[1] Includes flotation middling returned to the classifiers for dilution water

The circulating load maintained in the secondary circuit averages 170 per cent.

Rod loads are maintained at 18 tons in all mills, both primary and secondary. Three-inch rods are used exclusively in the primary mills, while those used in the secondaries are 2-1/2 inch. Scrap loss from the 3-inch rods averages 8 to 10 per cent, and slightly higher in the case of 2-1/2 inch rods. All rods are manufactured and furnished by the Compania Fundidora y Acero de Monterrey, of Monterrey, N. L., Mexico. Duties make the cost of imported rods prohibitive. The Mexican-made rods, however, seem to be as satisfactory as those purchased in the United States. A typical analysis of these rods is as follows:

	Per cent
Chromium ....:	0.25
Silicon ....:	0.14
Manganese ....:	0.62
Phosphorus ....:	0.03
Sulfur ....:	0.03
Carbon ....:	0.85

The consumption of these rods has averaged as shown in the table below, which includes scrap loss. The higher consumption for 1929 was brought about by a lower tonnage feed rate per hour, to maintain finer grinding.

Average consumption of rods

	Pounds per ton of concentrator feed			
	1926	1927	1928	1929
Primary mills......:	0.970	0.985	0.816	0.993
Secondary mills....:	1.047	1.047	1.145	1.192
Total ....:	2.017	2.032	1.961	2.185

All Marcy mill liners are cast in the local foundry. Shell liners, consisting of 10 sections 8 feet long, and 10 sections, 4 feet long, for a 6 by 12 foot mill, have a single wave down their center, and are 5 inches thick, exclusive of this ridge. There is no distinction in liners for right and left hand mills. For about a year some of the secondary mills were used with balls instead of rods, and during this time triple wave liners were used. When the use of balls was discontinued, the surplus stock of triple wave liners was used up in rod mills, at no observable detriment to their operation.

Feed end liners are cast in a single piece. Discharge end liners are also cast in a single piece to 18 or 30 inches diameter, depending upon whether it is for a primary or a secondary mill. Scrap loss on liners is about 15 per cent; the scrapped liners are returned to the foundry for remelting. The following figures on liner consumption include the discard weight:

Liner consumption

	Pounds per ton concentrator feed			
	1926	1927	1928	1929
Primary mills	0.31	0.25	0.25	0.24
Secondary mills	0.29	0.23	0.36	0.39
Total	0.60	0.48	0.61	0.63

Shell liners last slightly over a year; during 1928, 11 of the 12 operating mills were relined. These 11 sets of liners averaged a life of 9,500 operating hours, each handling 184,700 dry tons of ore. The longest life ever obtained from any set of locally cast liners was 15,541 operating hours, in which time 306,104 dry tons of ore were ground by that mill.

Regrinding of Middlings

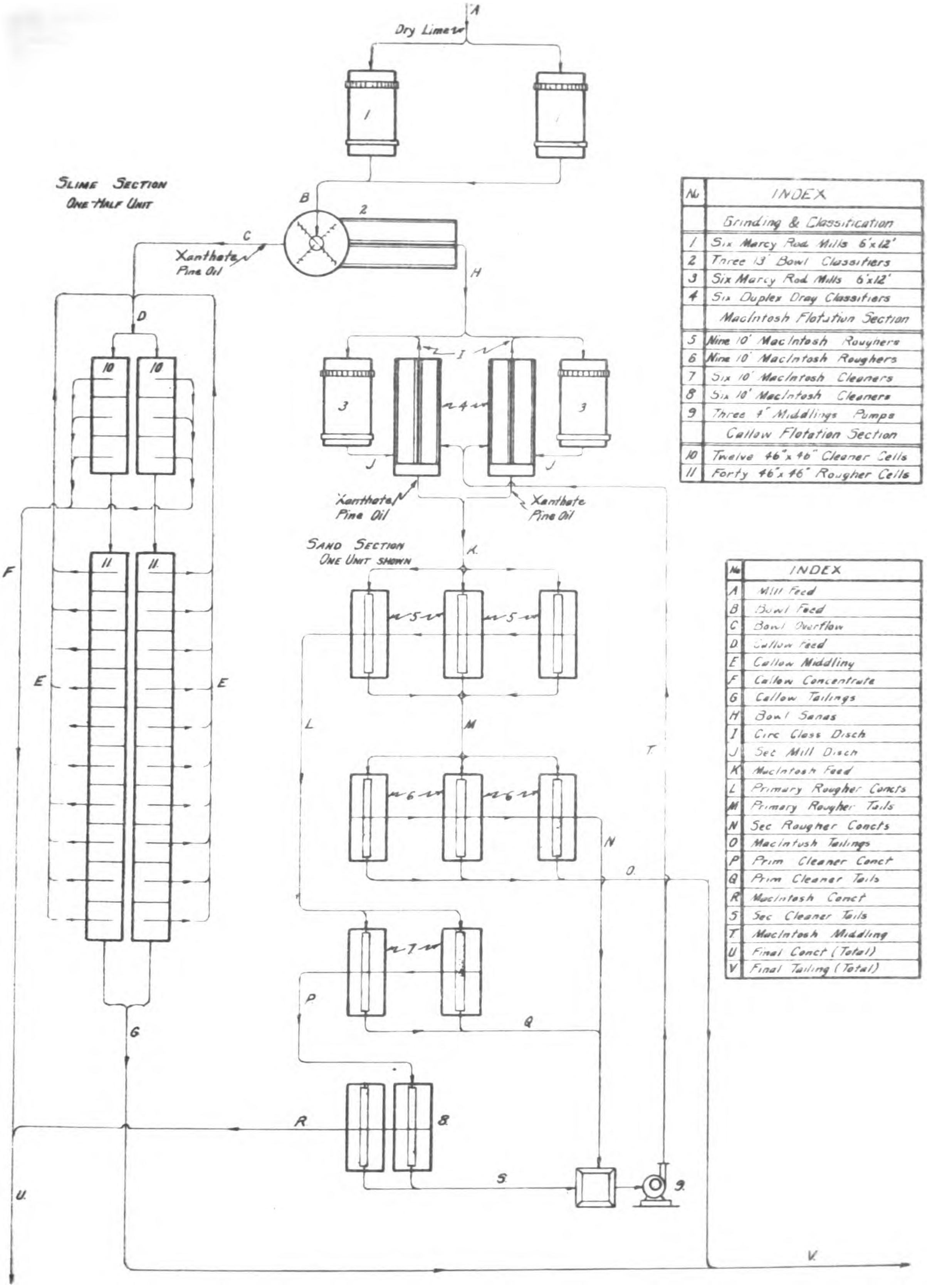
The only intermediate metallurgical product which is reground is the flotation middlings, and this operation is a very recent change and is still in the experimental stage. For that reason it must be dismissed in the present paper with the passing comment that promising results are being obtained.

Classification

In all, there are nine Dorr classifiers in the mill, of which three are of the bowl type.

The three bowl classifiers are 6 by 20 feet with 13-foot bowls. The reciprocating rakes are on a slope of 2 inches per foot, they make 22 strokes per minute, and are Tex-rope driven by individual motors. The bowl rakes are also Tex-rope driven by separate motor, through Reeves variable speed transmission. A bowl rake speed of five revolutions per minute is ordinarily maintained. In present practice a separation of 65 mesh is aimed at. It will be noted from the screen analyses previously given, that the sands contain 20.3 per cent of undersize. A heavy stream of backwash-water is used at the head of the reciprocating rakes.

The six circulating classifiers closing the secondary mill circuit are all 6 by 23 foot 4-inch and are set on a slope of 3-3/8 inches per foot. They are belt-driven from a countershaft and make 30 strokes per minute. Their sands, according to the screen analyses already given, carry 68.50 per cent of minus 48-mesh material. This apparently inferior classification is not literally true, however, as this product is heavily pyritic (carrying 26 per cent of iron, as compared with 10 per cent of iron in the original mill feed), and the selective grinding of the pyrite which thus results is metallurgically advantageous.



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	Grinding & Classification
1	Six Marcy Rod Mills 6'x12'
2	Three 13' Bowl Classifiers
3	Six Marcy Rod Mills 6'x12'
4	Six Duplex Dorr Classifiers
	MacIntosh Flotation Section
5	Nine 10' MacIntosh Roughers
6	Nine 10' MacIntosh Roughers
7	Six 10' MacIntosh Cleaners
8	Six 10' MacIntosh Cleaners
9	Three 4" Middlings Pumps
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10	Twelve 46"x46" Cleaner Cells
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A	Mill Feed
B	Bowl Feed
C	Bowl Overflow
D	Callow Feed
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H	Bowl Sands
I	Circ Class Disch
J	Sec Mill Disch
K	MacIntosh Feed
L	Primary Rougher Concts
M	Primary Rougher Tails
N	Sec Rougher Concts
O	MacIntosh Tailings
P	Prim Cleaner Conct
Q	Prim Cleaner Tails
R	MacIntosh Conct
S	Sec Cleaner Tails
T	MacIntosh Middling
U	Final Conct (Total)
V	Final Tailing (Total)

Figure 4.- Flow sheet for the last six months of 1929, Nacozari concentrator

Launder slopes in the mill circuit are as follows:

Launder slopes in the mill circuit

	Inches per foot	Per cent
Bowl sands to secondary mills........	4	
Circulating sands to secondary mills	4-1/2	
Flotation feed to flotation..........	7/16	
Primary rougher froth................	1-1/2	
Secondary rougher froth..............	1	
General mill tailings................		1.5

Flotation

A complete flow sheet of the flotation section is given in Figure 4, and a distribution of the tonnage, copper, iron, silver, and "insoluble" in Table 1.

The MacIntosh flotation machine was originated and developed at Nacozari to meet conditions resulting from the changed physical character of tailings after the development of selective flotation. While the primary objective of the machine was to keep heavy sands and pyrite moving freely through the circuit, it was immediately perceived that the accomplishment of this purpose should automatically improve both mechanical and metallurgical efficiencies. The anticipated advantages did indeed materialize. Whereas the Callow machines, in which the entire mill feed was treated for nearly three years, required 3.75 kw.h per ton of ore for blower air, the MacIntosh machines replacing them operated under identical conditions on the same feed and the same tonnage rate at an average power consumption of only 0.55 kw.h per ton, including power both for air and for driving rotors. At the same time, metallurgy was very materially improved.

After the period of development and probation in 1924-1925, two of the three 1,000-ton Callow units were torn out in 1926, and in the space thus made available were put three 1,000-ton MacIntosh units. As already noted, the third Callow unit was left idle, but was not torn out.

During the MacIntosh development period, various rotor and cell dimensions had been tried, and the new installation was standardized with machines 10 feet long, 38 inches wide at the overflow surface, and with a centerline depth of 32 inches. Rotors 9 inches in diameter were decided upon. These have an end clearance inside the cell of 4 inches at each end. Each rotor carries a longitudinal scraper whose sweep extends 2-1/2 inches outward from the rotor, so that the porous medium has no opportunity to rub the sides of the machine. A very important function of this scraper is to pick up any coarse material which ordinarily would tend to settle, and cast it over the zone of maximum aeration.

Each 1,000-ton unit consists of ten 10-foot machines, there being six roughers and four cleaners. The feed, now consisting of the reground bowl sands, and comprising about two-thirds of the total mill feed, is divided from a distributor to three primary roughers. These make concentrates of about 15 per cent copper and tailings of about 0.25 per cent copper. The tailings of each of these pass to a corresponding secondary rougher, which makes the final unit tailings of 0.12 per cent copper, but in so doing produces low-grade secondary concentrates averaging not over 4 per cent copper. In the original flow sheet the primary and secondary rougher concentrates were joined to pass into the cleaner circuit, but in 1928 this was changed in such a way that the secondary rougher concentrates went directly to the middlings sump for return to the head of the circuit, leaving only the primary rougher concentrates to go through the cleaner circuit. More recently the secondary rougher concentrates have been considered to be true middlings, and those concentrates from all three MacIntosh units are now being taken experimentally to a thickener, where they are being thickened to a sufficient density to permit their return directly into the secondary mills for some additional grinding. Prior to putting middlings regrinding into effect, the primary rougher concentrates, at 15 per cent copper, was sent to the primary cleaners, which made a tailings of about 0.80 per cent copper and stepped the concentrates up to 20 per cent copper. These concentrates then went to the one secondary cleaner, which is set immediately adjacent to the tertiary cleaner and about 4 inches above it, so that the secondary cleaner, blinded on one side, overflowed directly into the tertiary, which also overflowed on one side only. The secondary and tertiary cleaners stepped the grade up about 4 per cent and 2 per cent, respectively, giving a final grade of 26 per cent copper as against the theoretical of 34.63 per cent for chalcopyrite. Concentrates have been held as high as 28 per cent to 30 per cent for periods of a month or more, but for coarse feed this is not economical without provision for regrinding of middlings.

The tertiary cleaner tailings have been known to go as high as 14 per cent, and in such a case the limit of cleaning without intermediate regrinding was plainly being overstepped. The average grade of flotation return middlings, including all returned products, has usually averaged about the same as the assay of the original new feed. All was brought together in a common sump below the cleaners and pumped back to the circulating classifiers to be used as dilution water, since it averages only about 10 per cent of solids. Since it has averaged 85 per cent minus 200 mesh for many years, the classifiers had never been able to take out any of the solids for regrinding, especially since the same material had already overflowed these same classifiers in the first instance, before going to flotation originally.

Toward the latter part of 1928, a tendency toward increased difficulty in the production of high-grade concentrates began to be apparent, and this trend became more pronounced in 1929, due to a somewhat finer dissemination of chalcopyrite in the ore of the lower mine levels, which are now producing a larger share of the mine output. During the latter part of 1929 many laboratory flotation tests were run to determine if the direct return of flotation middlings to the head of the flotation circuit was metallurgically sound, and contrary to apparent conditions of some three or four years previous, it was found not to be. The regrinding of middlings was accordingly applied to the mill circuit in February, 1930. This has been too recent an innovation for the present paper to consider in terms of general mill results. None of the data herein presented take that change into account.

Table 1.--Distribution of tonnage, copper, iron, insoluble, and silver in flotation circuit for an average milling during the last six months of 1929

Reference letter, flow sheet Figure 4	Product	Weight, dry tons	Copper			Iron		
			Total pounds	Assay per cent	Per cent of input[1]	Total tons	Assay per cent	Per cent of input[1]
A	Mill feed	2,500.00	135,500	2.71	100.00	261.50	10.46	100.00
C	Bowl classifier overflow	779.60	45,695	2.93	33.72	65.03	8.34	24.88
D	Callow feed	944.88	59,612	3.15	43.99	84.28	8.91	32.24
E	Callow middlings	165.28	13,917	4.21	10.27	19.25	11.65	7.36
F	Callow concentrates	98.21	44,136	22.47	32.57	23.77	24.20	9.09
G	Callow tailings	681.39	1,559	0.11	1.15	41.26	6.06	15.79
H	Bowl classifier sands	1,720.40	89,805	2.61	66.28	196.47	11.42	75.12
I	Circulating classifier discharge	2,867.39	98,636	1.72	72.79	736.06	25.67	281.49
J	Secondary mill discharge	4,567.79	188,441	2.05	139.07	932.53	20.33	356.61
K	MacIntosh feed	2,031.19	109,882	2.70	81.10	243.34	11.98	93.04
T	MacIntosh middlings	310.79	20,077	3.23	14.82	46.87	15.08	17.92
R	MacIntosh concentrates	181.16	85,762	23.67	63.30	51.47	28.43	19.68
O	MacIntosh tailings	1,539.24	4,043	0.13	2.98	145.00	9.42	55.44
A	Mill feed	2,500.00	135,500	2.71	100.00	261.50	10.46	100.00
U	Total concentrates	279.37	129,898	23.248	95.87	75.24	26.93	28.77
V	Total tailings	2,220.63	5,602	0.126	4.13	186.26	8.39	71.23

[1] "Input" refers to total mill feed.

Table 1.--Distribution of tonnage, copper, iron, insoluble, and silver in flotation circuit for an average milling during the last six months of 1929. (cont.)

Reference letter flow sheet Figure 4	Product	Insoluble			Silver		
		Total tons	Assay, per cent	Per cent of input[1]	Total tons	Assay, ounces per ton	Per cent of input[1]
A	Mill feed	1,677.50	67.10	100.00	1,857.60	0.74	100.00
C	Bowl classifier overflow	537.85	68.99	32.06	786.92	1.01	42.36
D	Callow feed	637.35	67.45	37.99	1,203.43	1.27	64.78
E	Callow middlings	99.50	60.20	5.93	416.51	2.52	22.42
F	Callow concentrates	19.70	20.06	1.17	705.15	7.18	37.96
G	Callow tailings	518.15	76.05	30.89	81.77	0.12	4.40
H	Bowl classifier sands	1,139.65	66.24	67.94	1,070.68	0.62	57.64
I	Circulating classifier discharge	1,122.87	39.16	66.93	2,121.87	0.74	114.23
J	Secondary mill discharge	2,262.52	49.32	134.87	3,192.55	0.70	171.87
K	MacIntosh feed	1,319.22	64.95	78.64	1,496.46	0.74	80.56
T	MacIntosh middlings	179.57	57.78	10.70	425.78	1.37	22.92
R	MacIntosh concentrates	22.55	12.45	1.35	872.65	4.82	46.98
O	MacIntosh tailings	1,117.10	72.57	66.59	198.03	0.13	10.66
A	Mill feed	1,677.50	67.10	100.00	1,857.60	0.77	100.00
U	Total concentrates	42.25	15.12	2.52	1,577.80	5.80	84.94
V	Total tailings	1,635.25	73.64	97.48	279.80	0.13	15.06

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[1] "Input" refers to total mill feed.

When, in July, 1929, separate slime treatment was adopted, the idle Callow section was adapted to the purpose for no other reason than that the machines and launder system were in place and could be reconditioned at minimum expense. Treating slimes only, with 87 per cent of their feed minus 200 mesh, the Callow machines do not encounter their old difficulty with settling sands to nearly the same extent as formerly, although they still require punching out once or twice per shift. Their power consumption still averages 3.75 kw.h. per ton, just as formerly, and it is anticipated that they will be replaced by a more economical type of machine in reasonable time.

A few of the MacIntosh machines in the plant are individually equipped with Bailey air meters, graduated in terms of cubic feet of air used per hour. These motors confirm other elaborate and accurate air tests which have been made from time to time, indicating that the MacIntosh machines, of the size already stated, use an average of 6,000 cubic feet of air per hour each, or 100 cubic feet per minute per machine. For the 30 machines operating, this is 3,000 cubic feet per minute, and the entire amount is furnished by one No. 5-1/2 Roots blower, drawing an average of 55-horsepower. This blower furnished all of the flotation air required by the entire mill during the three years that the MacIntosh machines were treating the entire mill feed.

Previous to the change to MacIntosh machines, each 1,000-ton Callow unit required one No. 7 Roots blower driven by a 150-horsepower motor, so that three of these blowers were running to supply air for the three Callow units. These blowers were all shut down, and two of them were sold and removed from the mill when the change to MacIntosh machines was made. Now that one unit of Callows is again running, the mill requirements for flotation air are met by one No. 5-1/2 and one No. 7 Roots blower.

The porous medium used for the Callow machines is 2-ply canvas, quilted in 1-inch squares. For a time, similar but unquilted canvas was used as the porous rotor covering for the MacIntosh cells. Under such conditions the Callows operated at 4 pounds pressure per square inch and the MacIntoshes at about 3 pounds.

For the past two years, however, perforated gum rubber coverings have been used on the MacIntosh rotors. These are also a local development. They are high-grade gum rubber stock and have definite holes through them, 0.01 inch in actual, clear diameter, there being 180 such holes per square inch. These are rather more porous than canvas - in fact, when the sheets are held up to the light, one is able to see through them and distinguish objects on the other side. By means of a locally designed "porosity meter" the actual porosity of any medium is readily determined, and it has been found that these blankets pass as much air at 1 pound pressure as will canvas at 4 pounds pressure. Indeed, in practice they are able to operate at very low pressures, yet, as it has proved, without danger of back-filtration of solids. With these rubber blankets, the MacIntosh machines are operated at about 1-3/8 pounds pressure per square inch, or just sufficient to give a comfortable margin in overcoming the hydrostatic head of the pulp in the machine.

The average life of canvas flotation blankets at Nacozari is about 90 days. The rubber blankets last from six months to a year. Two are running which are more than a year old.

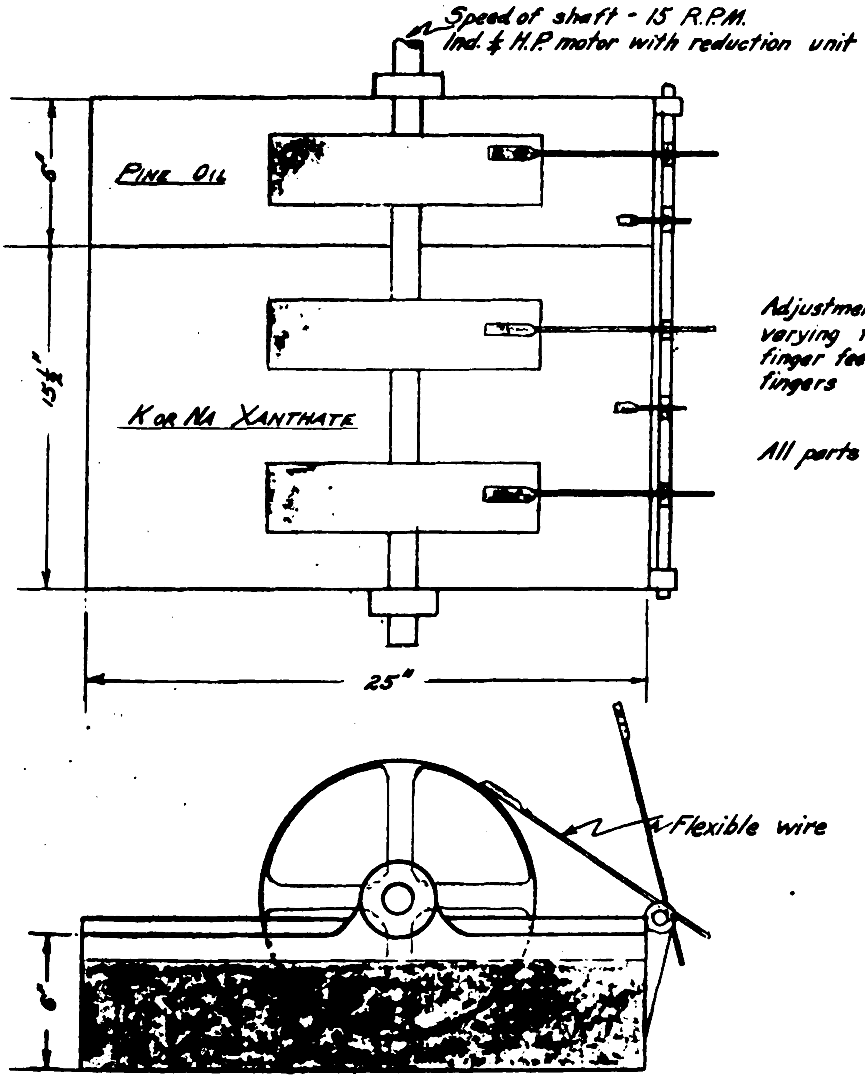
Flotation Reagents

Since the beginning of operations, steam-distilled pine oil has been, and continues to be, the frothing reagent used. On the MacIntosh sections, this is added to the circulating classifier overflow, and has about 30 seconds of mixing action in the feed launder before entering the flotation machines. Ethyl xanthate has been the collector reagent added from 1925 to the present time. For about two years the potassium salt was used, but more recently the sodium salt has been found to possess satisfactory purity and stability. It is fed in 6 per cent water solution, and the consumption on the MacIntosh sections averages 0.03 pounds of the solid reagent per ton of ore. The pine oil and xanthate are both fed at the same place—in fact, from separate compartments of the same feeder. For the slime section, the pine oil and xanthate are both fed from a similar feeder into the trunk launder carrying the general slime feed, and about one minute of mixing action is available. Due to the higher content of oxide copper in the slime, and probably also to the greater mineral surface, the consumption of xanthate on the slime section is slightly higher, and averages 0.04 pounds per ton of ore treated. Neither xanthate nor pine oil is added at any other place than to the primary rougher feed of the individual sections.

For Nacozari mill feed, certain of the nitrogen reagents have been found to yield better metallurgy than any of the sulphur reagents such as the xanthates or the thiophosphates. The nitrogen reagents possess the two virtues of being able to float coarser mineral and of showing no tendency to float free pyrite in an alkaline circuit. As against this, however, the nitrogen reagents are not as effective as the thio reagents in the recovery of oxide copper. Several mill-scale tests have been run which indicate tailings equal to, or slightly better than, those yield by xanthate and at the same time the grade of concentrates has appeared to be higher by about 2 per cent copper.

All lime used in the mill is fed in the dry state from a single feeder of the ratchet-and-pawl type upon the conveyor system leading from the fine ore bins to the primary mills. No lime or milk-of-lime is added at any other place in the metallurgical circuit. Prior to the adoption of separate slime treatment, the consumption of dry burned lime averaged 5.5 pounds per dry ton of ore treated. Since that time, the consumption has varied between 3 and 4 pounds. The grade of lime used averages about 60 per cent available CaO. On the present flow sheet, the tailings water of the three MacIntosh sections titrates about 0.10 pound CaO per ton of water; from the Callow slime section, the titration averages about 0.35 pound per ton, the larger excess at that point being due to the tendency for undissolved lime to overflow the bowls. A milk-of-lime system may eventually replace the present dry-lime feeder.

Lime is controlled by hourly titrations of unit tailings water with standard sulphuric acid solution. Determinations of pH values of mill solutions have been made at various times, but no direct metallurgical relationship has been established, as applied to present practice. A continuous, recording, graphic conductivity meter was purchased in 1924, and this has been found very satisfactory as a means of securing continuous graphs of alkalinity variations.


Speed of shaft - 15 R.P.M.
Ind. ¼ H.P. motor with reduction unit
Pine Oil
K or Na Xanthate
6"
15¼"
25"
Flexible wire
6"

Adjustments for consumption are made by varying the contact angle between the finger feeder and pulley - or by additional fingers

All parts cast and assembled locally

Figure 5.- Liquid reagent feeder

It is not ordinarily used, however, except for special test purposes, though it could readily be made to actuate and regulate the lime feeder mechanism.

A sketch of the reagent feeder used for xanthate solution and pine oil is given in Figure 5.

Separate Treatment of Primary Slimes

As already described, the separate treatment of primary slimes has been in effect since July, 1929, and has resulted in a very definite metallurgical improvement. The product treated separately is not true primary slimes in the sense of containing only the original fines of the mine ore. The product does indeed contain practically all of the original mine fines, but in addition it contains the minus 65-mesh material produced by the primary mills. The advantage of primary slime treatment appears to have been governed by chemical factors, rather than to have had anything to do with the flotation of "sized mineral," if the colliodal aspect of the situation be included in the chemical category.

The following table, giving the essential data of separate slime treatment, expresses the averages for the months of July to December, inclusive, 1929:

Average results of sand and slime sections
July - December, 1929

	Slime section	Sand section	Total mill
Weight of total mill feed, per cent...	30.70	69.30	100.00
Analyses of feed:			
Total copper, per cent.......	2.93	2.60	2.70
Oxide copper, per cent.... ..	0.110	0.039	0.061
Iron, per cent...............	8.37	11.80	10.75
Insoluble, per cent..........	69.12	66.13	67.05
Analyses of concentrates:			
Total copper, per cent.......	22.33	23.48	23.08
Iron, per cent...............	24.15	28.56	27.03
Insoluble, per cent..........	19.94	12.50	15.08
Analyses of tailings:			
Total copper, per cent.......	0.12	0.13	0.127
Oxide copper, per cent.......	0.050	0.035	0.040
Iron, per cent...............	6.08	9.82	8.69
Insoluble, per cent..........	76.26	72.48	73.62
Recovery:			
Total copper, per cent.......	96.55	95.51	95.86
Oxide copper, per cent.......	60.32	18.17	41.10
Iron, per cent...............	36.57	25.63	28.25
Insoluble, per cent..........	3.66	2.00	2.53
Concentration ratio....................	7.89	9.44	8.90

The following table gives the mill total copper recovery by individual mesh sizes of flotation feed for six months preceding the introduction of separate slime treatment, in comparison with a four months average period throughout which separate treatment was regularly in effect and reflects the benefit to recovery in the two extreme sizes.

Comparison of copper recovery by ordinary and separate treatment

	Per cent recovery of total copper in feed by individual mesh sizes					
	+ 65	-65+100	-100+150	-150+200	-200	Total
Ordinary treatment	81.11	93.95	97.41	98.37	94.40	93.56
Separate treatment:						
Sand..........	88.51	93.31	97.95	98.76	97.00	95.51
Slime.........	67.70	87.89	96.07	98.21	97.16	96.55
Total.........	87.93	92.92	97.70	98.66	97.08	95.87

Conveying

The Nacozari concentrator has four pan conveyors or pan feeders; four belt conveyors in the crushing plant; nine other conveyors handling crushed ore; two conveyors handling concentrates; and one bucket elevator handling concentrates.

The four bottom openings of the coarse ore bin are each discharged by a Webster 48-inch by 14-foot 6-inch pan conveyor. These handle mine-run ore, up to 16-inch size, at the rate of 95 tons per hour each, travelling 7.1 feet per minute. The flights are covered with 2-inch white oak, which is renewed about once every three years. The ore travel is on an upward slope of 17°. These conveyors are driven by Chabelco chain and sprocket through a clutch.

In the crushing plant, conveyors 1 and 3 each receive the discharge of a No. 8 McCully crusher, together with by-passed fines, at a maximum size of 4 inches. These travel 200 feet per minute on a 20° upward slope and are 49.5 feet long between centers. They handle 95 tons per hour, each. They are 6-ply belts, 36 inches wide.

Conveyors 2 and 4 in the crushing plant are also 6-ply belts, 36 inches wide, 19 feet long, center to center. They handle the Telsmith crusher discharge, at a maximum size of 3 inches, to the Symons crushers, together with by-passed fines, at a rate of 55 tons per hour. These belts make 200 feet per minute, and are on an up-slope of 20°.

The entire crushing plant output of ore, 190 tons per hour at a maximum size of 1-3/4 inch, joins upon conveyor 5 and is taken to the fine ore bin.

This belt is 327 feet long, center to center, is 24 inches wide, 5-ply, and travels 450 feet per minute. It works horizontally for a distance of 30 feet, and the remainder of its length is on a 16° slope.

Conveyor 6 is horizontal belt across the fine ore bin. It takes the total load of conveyor 5 and discharges over a Stephens-Adamson automatic, movable belt tripper. This belt is 24 inches wide, 5-ply, 177 feet long between centers, and travels 450 feet per minute.

Conveyors 7 and 8 are parallel horizontal belts beneath the fine ore bin, and each picks up the discharge of 17 Webster 18-inch apron feeders. These belts each handle 55 tons per hour of mill feed at a maximum size of 1-3/4 inch. Both are 5-ply belts, 175 feet long between centers, 20 inches wide, and travel 208 feet per minute.

Conveyors 9 and 10 are parallel belts, at right angles to, and receiving the discharge of, conveyors 7 and 8 respectively. They have the same characteristics of conveyors 7 and 8 in every way, except that they are on a 20° slope and are 48 feet long between centers.

Conveyors 11, 12, and 13 are respectively the feed belts for the three mill sections. They also have the same characteristics as conveyors 9 and 10, except for distance of travel. The three lengths are respectively 113 feet, 46 feet, and 45 feet, between centers.

Power for ore conveying averages 0.53 kw.h. per ton of ore, for the entire system above detailed. There are no elevators handling ore, and all elevation is gained by the inclination of the conveyors. However, the sum of these inclinations is the equivalent of elevating the entire mill feed a vertical distance of 125 feet.

Conveyors 14 and 15 until recently handled that portion of the flotation concentrates which was customarily settled and dewatered by gravity. As all concentrates are now filtered, these conveyors stand by for emergency only.

There is only one bucket elevator operating in the Nacozari concentrator (although a similar elevator stands by as a spare). This is equipped with an 18-inch belt, 115 feet total belt length, making a speed of 364 feet per minute, with buckets spaced 32 inches apart. It handles such flotation concentrates as are thickened, this amounting to 170 tons of solids per day, at 60 per cent solids. The average power consumption is 2.80-horsepower constant input.

Prior to 1926, the lay-out of the flotation plant was such that the middlings returned to the head of the circuit consisted of rougher concentrates only. Due to the frothy character of this product, it was all handled in air lifts. With the new MacIntosh plant lay-out, however, provision was made to return cleaner tailings rather than rougher concentrates, and this product is now all handled by centrifugal pumps. With the restoration of one section of

Callows in 1929, pumps were put in to handle their middlings also; this continues to be rougher froth as formerly. All pumps for middlings service, and for tailings elevation as below described, are of a standard design and of the standard size of 4-inch, with 18-inch runner. These pumps, designed and patented by H. Littlewood, concentrator master mechanic, are cast in the local foundry; they are of hard white iron, and their design is such that no machining is required. Runners are keyed to the shafts. These pumps are made very cheaply - at about 15 per cent of the price at which equivalent pumps could be imported - and the local practice is to scrap them and put in new pumps as soon as they fall below a stipulated standard of efficiency from wear. The worn parts are, of course, returned to the foundry as scrap.

Beginning in 1926, it has been necessary to pump the general mill tailings into the tailings disposal basin. This has been at gradually increasing height; the vertical pumping lift is now 67 feet high and is handled in two stages. The following table gives the essential data on pulp elevation by means of centrifugal pumps.

Pumping data for clear water are given under the subject of water.

Data on Pulp Elevation by Means of Centrifugal Pumps

	Flotation middlings		General mill tailings	
	Callow[1]	MacIntosh	Lower stage	Upper stage
Number of pumps running	2	3	2	2
Number of pumps standing by	1	3	2	1
Size of pumps, inches	4	4	4	4
Diameter of runners, inches	18	18	18	18
Speed, revolutions per minute	690	720	830	630
Horsepower of motors	20	10	40	20
Average power input, kw	9.8	10.2	21.3	12.9
Vertical lift	45'5"	35'	44'7"	21'7"
Solids in pulp, per cent	7.6	13.1	19.1	19.0
Specific gravity of pulp	1.14	1.15	1.18	1.18
Gallons pulp per minute	154	112	852	852
Average efficiency, per cent	25 - 35	30 - 45	45 - 55	45 - 55

[1] Froth.

DEWATERING AND HANDLING OF CONCENTRATES

The Nacozari concentrator produces but one class of concentrates--flotation concentrates, but this product comes from two sources - the one Callow slime unit and the sum of three MacIntosh sand units.

Because of its relative coarseness, the dewatering of flotation concentrates at Nacozari has always been attended by some difficulty because of the settling and packing of coarse mineral, with consequent breakage of thickener rakes and filter agitators. Because of this condition, it has not even yet been found feasible to send all flotation concentrates directly to thickeners. Until very recently it was the practice to table all concentrates and thus to separate out a portion of the coarser mineral which could be settled and decanted by gravity in bins somewhat of the Blaisdell type. Filtering 70 per cent of the total concentrates production (consisting of the finer material) and settling the remaining 30 per cent, gave an average moisture for the total of 12.50 per cent.

At the present time, however, means has been found to filter the entire concentrates production, and the average moisture of the entire tonnage has thereby been reduced to 10 per cent. It had been noted that the froth of the final flotation cleaner cells was averaging above 40 per cent of solids, and it was found that, with a little care, this could be maintained at approximately 50 per cent of solids. This was deemed to be of sufficient density to send directly to the filter, and launder connections for the purpose were put in from two of the MacIntosh units directly to the filter. The third MacIntosh unit concentrates, together with the concentrates of the Callow slime section, are taken to a thickener, as they together do not contain enough coarse material to exceed permissible limits. This joint product, thickened to 60 per cent of solids, is taken to the same 14 by 14-foot Oliver filter which receives the total concentrates of the two above MacIntosh units directly, thus making a total filter feed of about 55 per cent of solids. The one filter, operating thus, is able to handle the entire mill production of concentrates, amounting to about 275 tons of solids per 24 hours. Agitators in the filter are of the oscillating type, with a 2-foot stroke, and they make 26 strokes per minute. The agitator is driven by a motor separate from that driving the drum. The drum makes one revolution in 11 minutes.

The filter is covered with No. 17-ST canvas twill as furnished by the Oliver Co., or the Filter Fabrics Co. of Salt Lake City. This is the same material as formerly was used for covering MacIntosh rotors, prior to the time of developing perforated rubber. The filter covering is backed with 1-ply of 10-ounce burlap, and is wrapped with No. 12 Premier spring wire, spaced 3/4 inch between turns. The life of the blanket averages four months.

The pulp filtered is barely alkaline to phenolphthalein, varying from 8.3 to about 9.0 pH. The temperature ranges from about 40° F. in winter to 65° F. in summer.

Formerly, Nash Hy-tor wet-vacuum pumps were used to produce vacuum for the filters and the gravity bins, there being one such pump for each filter, and one for each bank of eight bins. In 1929, however, these were all replaced by one dry-vacuum pump. In former years, when flotation middlings were being airlifted back to the head of flotation, two 20-pound duplex Ingersoll-Rand XRB-1 compressors were used in this service, and they were rendered idle when centrifugal pumps replaced the air-lifts. It is one of these compressors which is now used as the dry-vacuum pump, the compression having been reversed. Only one cylinder is used. This machine has proved to be very satisfactory for the purpose and gives 20 to 23 inches of vacuum at the filters, with an average input of 38 kw.

With a freight rate on concentrate to the smelter of about $2.40 per ton of material hauled, the use of a drier to work upon filter cake has been given careful consideration, but profit from such an additional operation has not been found demonstrable, and no such device is used. Indeed, as the concentrates are several days in transit to the smelter, solar evaporation is sufficient to make possible dust loss a factor to be guarded against.

The amount of water recovered for reuse is tabulated in the water balance given in Table 2.

DEWATERING AND DISPOSAL OF TAILINGS

All mill tailings join in a common launder at the maximum elevation practicable below the flotation floor and, without thickening, are thus transported a distance of about three-fourths of a mile to the tailings disposal basin, which is a side canyon tributary to the Nacozari River. The flume goes over rough terrain, around curves and through some small cuts, and is supported on trestles across gulches where necessary. For 1,290 feet of the distance, it passes through a 5 by 7-foot tunnel.

During 1929 an average of 2,295 dry tons of tailings were produced per 24 hours of actual running time. The average screen analysis was as follows:

Average screen analysis of tailings

	Per cent weight retained	
	This mesh	Cumulative
Plus 35	1.80	1.80
Plus 48	5.31	7.11
Plus 65	7.16	14.27
Plus 100	14.93	29.20
Plus 150	8.10	37.30
Plus 200	4.48	41.78
Minus 200	58.22	100.00

The entire tailings launder is on a grade of 1.5 per cent, this being the minimum slope that will satisfactorily carry the tailings with the screen analysis above shown. On straight stretches, a grade of 1.2 per cent will carry, though with the danger of choking in the event of surges of oversize. This launder, made of 2 by 12 inch lumber, has 22 inches inside width in the clear and

10 inches inside depth. The tailing stream ordinarily runs 4 to 5 inches deep, averaging 19 per cent of solids. The stream velocity is 9.2 feet per second, corrected for drag. The launder is lined on both sides and the bottom with 1-inch wood, the bottom lining being crosswise of the stream to prevent channelling with the grain. The sections of the launder are of a standard length of 14 feet, which is also the spacing between bents on the trestles. At each joint, and at two other points in each 14-foot section, the launder is tied together by circumscribing quadrilateral ties of 2 by 4 inch lumber.

The present tailings disposal site was a virgin area when the mill began operating in 1922. A sand dam, made up of the coarser portion of the tailings, was gradually thrown across the mouth of the canyon by dropping this material through spigots or automatic thickening cones, by gravity, from a trestle of sufficient height. New trestles were built across the face at intervals as the height of the dam increased, and the process was repeated, each addition being stepped back to conform with the angle of repose. The water and finer sands were conveyed by launder upstream from the dam so formed, and there impounded by the dam. A vertical concrete spill-way chimney was provided inside of the sand dam, with a bottom outlet through a tunnel in the toe rock at the level of the canyon floor. Four-foot additions to this chimney were poured as necessary. This chimney served chiefly to carry off flood waters or excess water not returned to the mill, and to control the water level of the impoundage lake.

As the tailings pass through a tunnel enroute from the mill to the pond, it was necessary to begin elevating tailings in 1926 when the basin filled up the available space to the tunnel level and under such conditions to do this pumping at the tailings pond rather than at the mill to avoid the expense of constructing a new tunnel.

Four-inch Littlewood sand pumps were put in for this service, as already noted, and they continue in use to the present time. A new tunnel, 100 feet above the old location, is now being constructed and will be in service in mid-1930. All tailings elevation pumps will then be located within the mill building. The 100-foot lift will be negotiated in two equal stages.

Due to setting back with each rise of the tailings dam, the vertical spillway chimney is now outside the face of the upper portion of the sand dam. An overflow ditch cut in the hillside now serves the same purpose and discharges into the top of the old chimney. The total height of the sand dam is now approximately 140 feet. At its crest it is 975 feet long. The dam has never been broken or "lost" despite the fact that the canyon in which it is located drains approximately 3 square miles. The heaviest rain yet recorded in the vicinity has been 6 inches in 24 hours.

At present, about 750 gallons of water per minute are being recovered from the tailings pond. This is taken out of the pond by a centrifugal pump mounted on skids at the shore line and is returned to the mill in a 12-inch redwood stave pipe line. At the mill this line discharges into the sump tanks which also serve the return water recovered from concentrates thickening operations, and is thence pumped back to the head of the circuit.

SAMPLING

The concentrator, as built in 1922, was equipped with complete automatic sampling system, the samplers all being actuated simultaneously by water pistons, centrally controlled from a tipping box. All significant intraprocess products were sampled automatically on each unit. Contrary to practice in the former mill, however, no general heads sampling plant was provided, the sampling of the feed to the individual mill sections being depended upon for this. The sampling system remains essentially the same to the present time, except that of course simplification of flowsheet has eliminated a number of the samples formerly taken.

The weight of ore charged to the mill is accurately controlled by track scales located at the mine. These are periodically inspected by the scale department of the Southern Pacific Railway. In addition, each of the three mill sections is equipped with a Merrick Model "E" weightometer. These serve chiefly to give the distribution of the ore by units and by shifts.

All cars of concentrates shipped are also weighed by track scales as they leave the mill. These scales are similarly subject to rigid inspection. The cars are individually sampled by hand, by means of a slotted pipe. The mill weights and assays of concentrates shipped are used only as a check. Smelter settlement weights and assays govern in the final mill control accounts.

The tailings of the three MacIntosh units and the one Callow unit are individually sampled by shifts making 4 samples per shift or 12 tailings samples per day. In addition, the general mill tailings are sampled by shifts, giving three more or a total of 15 tailings samples per day. The unit tailings are titrated for copper colorimetrically and the general samples by the iodide method as a check. As a further check upon operations, the three MacIntosh units and the Callow unit are sampled and titrated for copper hourly for five hours of each day shift, making an additional 20 samples per day or 35 in all. Monthly composites of the general mill heads, tailings, and concentrates are sent to the Douglas laboratory each month for check assay and complete analysis. Under these conditions, there is sufficient assurance of the accuracy of the assay of tailings losses.

Under these circumstances the mill control data must be correct, for the weight of feed and concentrates is fixed by scale weights independent of the mill, thus also establishing the weight of tailings by difference. Concentrates assay being accepted as returned by the smelter, the heads assay as determined by nine samples daily must check the tailing assay as determined by 15 samples daily, to satisfy the relation Hh - Cc - Tt, where H, C, and T, are tonnages of heads, concentrates, and tailing, respectively, and h, c, and t their respective assays.

Any discrepancies which creep in, as well as certain small but unavoidable mechanical losses are reported as "unaccounted for loss or gain." The average annual loss or gain for the past several years has been as follows, expressed as percentage of total mill input of copper:

Year	Per cent
1925	0.03 loss
1926	0.13 gain
1927	0.13 loss
1928	0.09 gain
1929	0.11 loss
average	0.06 loss

For many years, a moisture determination was made upon a mill heads sample taken at the feed chutes of the primary mills. Recent increases in dust, however, have made it necessary to introduce water sprays at various points, and this rendered a moisture determination valueless. An arbitrary value, equal to the average of the three previous years, is now taken.

WATER SUPPLY

Practically all new water used in the mill comes from the Huacal Dam and Reservoir, located about 3 miles northeast of Nacozari. The dam, 100 feet high, lies across a narrow canyon at that point and impounds in the more open country above a total of slightly under one billion gallons of water, mostly from the run-off of the summer rainy season. The dam usually begins overflowing the spillways in July. Ordinarily the level drops to about 20 feet below the spillways before the rainy season of the following year. The dam site is at sufficient elevation that the water flows by gravity to the supply tank of 500,000 gallons capacity at the head of the mill, through a pipe line 2.91 miles long. Half the line is 8-inch riveted and ashpalt-dipped pipe; for the remaining distance, 10-inch pipe of the same kind is used. The line passes through two "break-pressure boxes" to hold the pressure upon the line within safe limits.

In emergency, water may also be taken from the sands of the Nacozari River by pumping, although the Huacal Dam furnishes an abundant supply.

An analysis of the water from these two sources is given in the table below, with similar data for reclaimed water. All water used in the Nacozari mill is quite uniform and clear and contains no salts of proved detriment to flotation. It is not subjected to any pretreatment, except as it meets lime in the mill circuits. It quickly throws down its carbonates and bicarbonates, however, at points of contact with the lime-containing reclaimed water and scaling up of lines at those points has been a matter of some concern and difficulty. The remedy is to so arrange the water circuit that the two classes of water do not some in contact in pipe lines, but this objective has not been wholly achieve

As noted in connection with concentrates handling and tailings disposal water is reclaimed in connection with both those operations. The distribution is as follows:

	Gallons per ton of ore milled
New..........	523.8
From tailings pond.................	426.1
From concentrates thickeners......	235.4
Total	1185.3

Detailed data on water consumption are shown in the water balance in Figure 6.

Following are typical analyses of the new water available for use in the Nacozari concentrator, in parts per million:

	Huacal Dam	Nacozari River
$NaCl$...............	--	22.80
$CaCl_2$..............	1.87	---
Na_2SO_4.............	--	32.10
$CaSO_4$..............	15.00	80.10
$Ca(HCO_3)_2$..........	110.00	263.00
$Mg(HCO_3)_2$..........	32.00	175.70
$Fe(HCO_3)_2$..........	5.27	17.50
SiO_2...............	11.00	28.00

During most of 1929, monthly composite analyses of the two classes of reclaimed water were regularly made. These averaged as follows for nine months:

Average of monthly composite analyses of the two classes of reclaimed water, for nine months

	Total dissolved solids	Organic and volatile	SiO_2	Fe_2O_3	CaO	MgO	SO_3	Cl
Dorr thickeners.	443.0	58.0	11.1	2.3	116.7	3.3	195.5	6.0
Tailings pond...	758.2	85.3	12.3	2.3	207.8	3.6	350.1	8.2

The mill return water pumps are located on the thickener level near the foot of the mill. Water from the tailings pond comes in at the same level. The two classes mix and are pumped to the head of the mill by two Aldrich triplex pumps, 14-inch diameter, 12-inch stroke. There are three return water storage tanks at the head of the mill, but most of the water is taken out of the line for use at the levels required, without being pumped to the elevation of the tanks.

Mechanical efficiency tests are made upon these pumps periodically, by measuring the power input and motor efficiency while the pump is filling a tank of known elevation and capacity in measured time. Following are data from such a test:

R. p. m....................	43.8
Gallons per minute.........	892.84
Mean lift, feet............	145.2
Net horsepower to pump.....	41.53
Per cent mechanical efficiency.................	78.97

Small quantities of water are picked up from various sources by 4-inch centrifugal pumps of the Littlewood type, and are so handled to the main return water pumps. The total power for water pumping for 1929 averaged 0.97 kw.h. per ton of ore. This, of course, is for reclaimed water only, as it is not necessary to pump any fresh water.

CONCENTRATOR RECOVERY

Complete chemical analyses of concentrates and tailings are given in Table 4; metallurgical data are shown in Table 5. Data upon the constitution of concentrates, as revealed by the microscope, are obtained chiefly by frequent observation of fields of the several mill products. Determination and microscopic analysis of bakelite-mounted specimens have also been made by the Douglas laboratory, but as remarked elsewhere in this paper, these data are not yet considered to be fully coordinated as a means of operating control.

A series of laboratory studies have been made at Nacozari to determine the "critical size of flotation"--that is, the size above which concentrator recoveries begin to fall off, even though the particle be pure chalcopyrite. By laboratory methods, the critical point for pure, free chalcopyrite was found to be 100-mesh size with xanthate as the collector. Under conditions paralleling those of the mill, recoveries rapidly decreased as the mesh size increased, until they had dropped practically to zero at 28-mesh size. This is fully borne out by the regular milling results, as shown in the table of recoveries by meshes already given. Under conditions of relatively low alkalinity, however, it is found that the chalcopyrite need not be ground free from gangue in order for it to float. Microscopic examination shows that for particles of 100-mesh size or finer, it is necessary only to have a minute showing of chalcopyrite exposed upon the surface of a particle in order for it to float, even though the bulk of the particle be barren gangue. To a lesser extent, particles coarser than 100-mesh size can also be floated if only they have chalcopyrite exposed. In high alkalinity on the other hand, particles of any mesh must be nearly pure chalcopyrite in order to float.

In the low alkalinity now maintained at Nacozari, the class of mixed grains which readily float in the rougher cells float nearly as well in the cleaner circuit and so appear in the final concentrates, diluting the latter with their attached gangue. With present grinding, such as shown by the screen analyses presented in Table 3, the mixed grain material goes to final concentrates or to final tailings depending, respectively, upon whether low or high alkalinity is used. In the absence of provision in 1929 for the regrinding of middlings, it was more profitable to throw the mixed grain material to concentrates, and that was the practice followed. To grind Nacozari mill feed to the point of, say, 90 per cent absolute libration of the chalcopyrite would require about 200-mesh grinding, the cost of which not only would be prohibitive, but would be unjustified, in view of the fact that exposure rather than liberation of the chalcopyrite is all that is necessary to effect a concentration of the mixed-grain material and to give a reduced bulk of material to be reground.

No direct measure of the effect of fineness of grinding upon grade of concentrates produced is available, because this is--as just stated--partially a function of alkalinity in directing mixed-grain material to concentrates or to tailings. The following table gives a comparison of the copper losses in the plus 100 and minus 100 mesh sizes for the last six months of 1929.

Comparison of copper losses in plus 100 and minus 100 mesh sizes for the last six months of 1929

	General mill tailings	
	Per cent weight	Per cent copper
Plus 100 mesh	31.40	0.210
Minus 100 mesh	68.60	0.086
Total	100.00	0.125

SAFETY

No article upon any Phelps Dodge concentrator plant would be complete without some mention of safety. For the past several years an unusually comprehensive campaign has been under way, with constantly increasing momentum. The results have been most gratifying, in point of reduction of accidents and hazards, and of improvement in the spirit of alertness and enthusiasm. With an average of 125 men working per day, there have been but two lost-time accidents in the Nacozari concentrator since September, 1925, and both of these were of a minor character.

COSTS

Table 6 gives concentrator operating costs and Table 7 gives a summary of operating costs in units of labor, power, and supplies, averaged for the years 1928 and 1929.

Table 3.--Screen sizes during period covering last six months of 1929

Screen sizes	Mill feed	Primary mill discharge	Bowl overflow			Bowl sands (Secondary mill feed)[1]						Total mill Concentrates	Total mill tailings
			Heads	Concentrates	Tailings	Secondary mill feed	Circulating class discharge	Circulating class discharge	Middlings return	Concentrates	Tailings		
Plus 1-1/2 inch ring	4.42	--	---	--	--	--	--	--	--	--	--	---	--
Plus 1-1/4 inch ring	5.71	--	---	---	--	--	--	--	--	--	--	---	--
Plus 1 inch ring....	8.64	--	---	--	--	--	--	--	--	--	--	---	--
Plus 3/4 inch ring..	10.19	--	---	--	--	--	--	--	--	--	--	---	--
Plus 1/2 inch ring..	11.79	--	---	--	--	--	--	--	--	--	--	---	--
Plus 3 mesh.........	11.03	--	---	--	--	--	--	--	--	--	--	---	--
Plus 4 mesh.........	7.07	--	---	--	--	---	--	--	--	--	--	---	---
Plus 6 mesh.........	6.95	0.35	---	---	--	0.50	--	--	--	--	--	---	---
Plus 8 mesh.........	4.65	0.93	---	--	--	1.18	--	--	--	--	--	---	---
Plus 10 mesh........	4.20	2.77	---	--	--	5.08	--	--	--	--	--	---	--
Plus 14 mesh........	3.89	7.57	---	--	--	10.48	--	--	--	--	--	---	--
Plus 20 mesh........	2.85	8.67	---	--	--	12.38	0.70	--	--	--	--	---	--
Plus 28 mesh........	2.89	11.82	---	--	--	16.70	3.90	--	--	--	--	---	--
Plus 35 mesh........	1.85	9.05	---	--	--	12.81	8.10	0.40	--	--	2.07	---	1.80
Plus 48 mesh........	1.71	8.52	0.73	--	0.95	11.87	18.80	5.30	2.40	2.88	6.50	2.00	5.31
Plus 65 mesh........	1.38	5.28	1.23	0.67	1.62	8.50	17.10	6.90	3.55	10.35	9.88	7.38	7.16
Plus 100 mesh.......	2.76	10.12	4.37	2.15	4.97	12.50	31.40	16.90	10.40	16.27	21.05	11.94	14.93
Plus 150 mesh.......	0.97	4.77	4.50	3.78	4.46	4.00	11.70	8.40	7.40	16.33	11.37	12.48	8.10
Plus 200 mesh.......	0.49	2.57	3.17	4.37	3.00	1.38	2.90	4.10	4.75	11.81	5.93	9.52	4.48
Minus 200 mesh......	6.96	27.58	86.00	89.03	85.00	2.62	5.40	58.00	71.50	42.36	43.20	56.68	58.22

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[1] See flow sheet, Figure 4, for distribution of primary mill discharge. The products of the bowl classifier are treated separately by flotation, and the concentrates and tailings are combined to form final mill products.

Table 4.--Average mill assays and recoveries for the last six months - 1929

Total mill	Weight, per cent	Assays: Per cent: Total Cu	Oxide Cu	Sulphide Cu	Fe	SiO_2	Al_2O_3	S	CaO	Rem.	Ounces per ton: Ag	Au
Average heads	100.00	2.70	0.05	2.65	10.5	48.7	13.7	9.2	2.0	13.2	0.77	0.003
Average concentrates	11.23	23.08	0.07	23.01	27.0	9.0	2.7	31.4	0.9	5.9	5.80	0.022
Average tailings	88.77	0.13	0.04	0.09	8.7	53.6	15.0	6.5	2.1	14.0	0.12	0.001

Metallurgical segregations - sand and slime section - last six months - 1929

	Slime section			Sand section			Total mill		
	Copper	Iron	Insoluble	Copper	Iron	Insoluble	Copper	Iron	Insoluble
Percentage of elements.									
Heads	2.93	8.37	69.12	2.60	11.80	66.13	2.70	10.75	67.05
Tailings	0.12	6.08	76.26	0.13	9.82	72.48	0.13	8.69	73.62
Concentrates	22.33	24.13	19.94	23.48	28.56	12.50	23.08	27.03	15.08
Tons of elements:									
Heads	3,634.61	10,378	85,701	7,287.10	33,026	185,064	10,921.71	43,404	270,765
Tailings	125.23	6,583	82,567	327.13	24,560	181,359	452.36	31,143	623,926
Concentrates	3,509.38	3,795	3,134	6,959.97	8,466	3,705	10,469.35	12,261	6,839
Pounds of copper:									
Heads	7,269,206	--	--	14,574,221	--		21,843,427	--	--
Tailings	250,468	--	--	654,272	--		904,740	--	--
Concentrates	7,018,738	--	--	13,919,949	--		20,938,687	--	--
Percentage extraction	96.55	36.57	3.66	95.51	25.63	2.02	95.86	28.25	2.53

Table 4.--Average mill assays and recoveries for the last six months - 1929. (cont.)

Metallurgical segregations - sand and slime section - last six months - 1929 (cont.)

100-mesh tailings material from monthly screen composite	Slime section		Sand section		Total mill	
	Per cent weight	Per cent copper	Per cent weight	Per cent copper	Per cent weight	Per cent copper
Plus 100-mesh	7.54	0.24	39.50	0.21	29.78	0.22
Minus 100-mesh	92.46	0.11	60.50	0.08	70.22	0.09
Total	100.00	0.12	100.00	0.13	100.00	0.13
Pounds CaO per ton of tailing solution	0.361		0.100		0.182	
Concentration ratio	7.89		9.44		8.90	

Table 5.--Metallurgical data for last six months of 1929

Total ore treated, tons	403,839.00
Days operated[1]	169.00
Hours operated per day[2]	------
Average ore treated per 24 hours actual operation of total mill[1], tons	2,516.10
Average sections running[2]	------
Average ore treated per 24 hours per section, tons	838.67
Total copper, concentrates, tons	45,359.55
Concentrates per 24 hours actual operation total mill, tons	282.87
Recovery of total copper, per cent	95.862
Recovery of sulphide copper, per cent	97.132
Recovery of oxide copper, per cent	33.401
Recovery of gold, per cent	75.000
Recovery of silver, per cent	84.940
Ratio of concentration	8.903
Water consumption, gallons per ton ore, total	1,193
Water consumption, gallons per ton ore, fresh	524
Water consumption, gallons per ton ore, return	669
Rod consumption per ton ore, primary mills, pounds	0.993
Rod consumption per ton ore, secondary mills, pounds	1.192
Liner consumption per ton ore, primary mills, pounds	0.24
Liner consumption per ton ore, secondary mills, pounds	0.39

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[1] Mill does not operate on Sundays or Mexican national holidays.

[2] Of the total 184 calendar days covered in this report, the concentrator actual operating time was equivalent to three units (the entire mill) being in operation for 87.151 per cent of the total possible calendar period.

Table 6.--Costs - years 1928 and 1929 - per dry ton ore

Note: Costs given below are for the total two year period

	Total labor	Power	Total supplies[1]	Grinding materials and reagents	Flotation royalty	Miscellaneous	Total
Crushing and conveying...	0.0262	0.0117	0.0236	--	--	--	$0.0615
Primary grinding.........	.0111	.0485	.0047	0.0416	--	--	.1059
Secondary grinding....	.0127	.0457	.0045	.0541	--	--	.1170
Flotation.....	.0105	.0100	.0084	.0667	0.0097	--	.1053
Concentrates handling....	.0089	.0042	.0039	--	--	--	.0170
Tailings......	.0144	.0071	.0061	--	--	--	.0276
Mill water....	.0073	.0086	.0020	--	--	--	.0179
Assaying and sampling....	.0098	.0007	.0032	--	--	--	.0137
Mill department expense	.0362	.0020	.0054	--	--	0.0035	.0471
Total......	0.1371	0.1385	0.0618	0.1624	0.0097	0.0035	$0.5130

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[1] The column "total supplies" does not include the cost of any grinding materials or flotation reagents.

Table 7.--Summary of costs in units of labor, power, and supplies

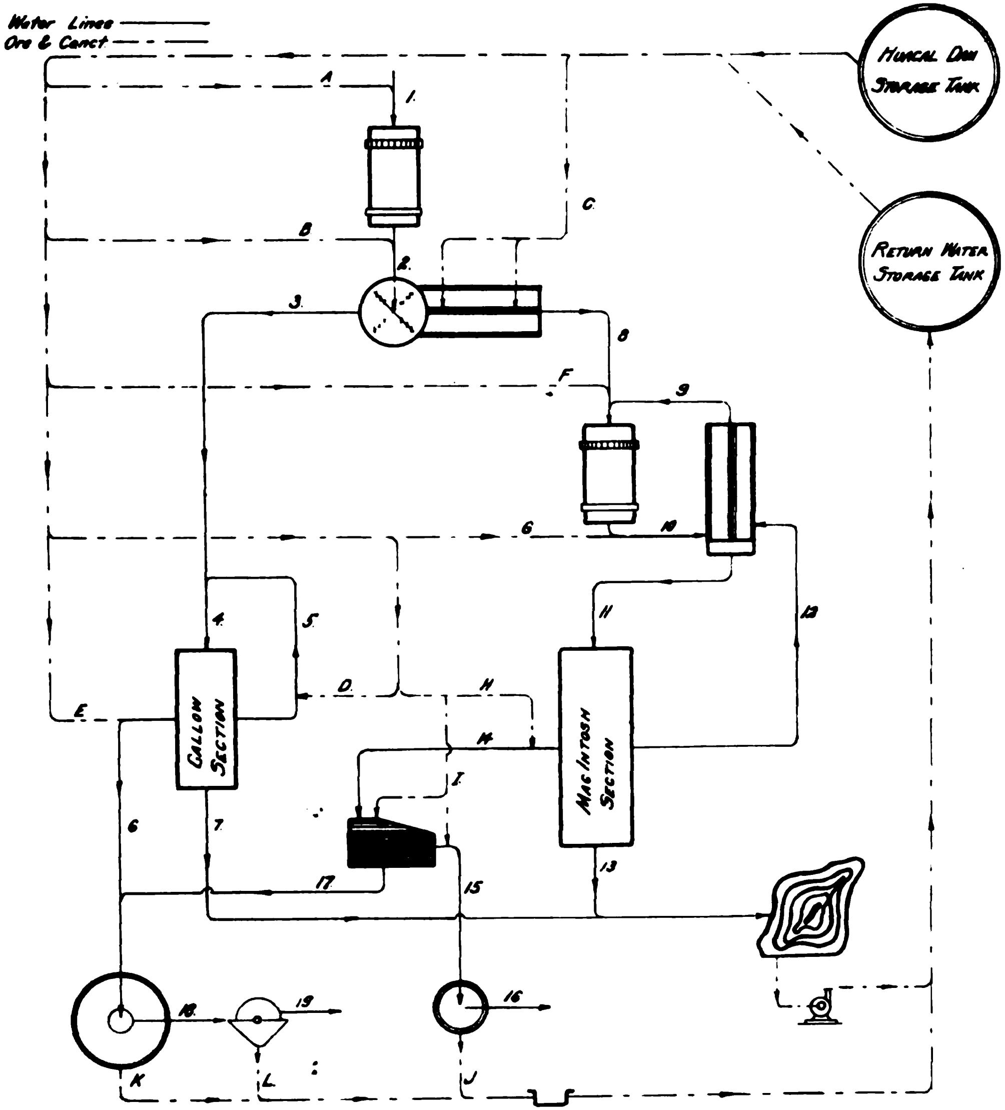
Period covered: Tons ore treated: 1,646,706
Average of years 1928 and 1929 Tons concentrates: 173,035

	Operation	Repairs	Total
A. Labor (tons per man per 8-hour shift)	30.06	54.32	19.35

Cost, dollars per ton of ore milled:	Total
Intermediate crushing..........	0.0251
Primary grinding...............	.0111
Secondary grinding.............	.0127
Classifying and conveying.......	.0011
Flotation......................	.0105
Concentrates handling..........	.0089
Tailings disposal..............	.0144
Water supply...................	.0073
Experimental charge............	.0038
Assaying and sampling..........	.0098
Miscellaneous..................	.0324
Total	0.1371

	Dollars per ton ore milled	Kw.h. per ton ore milled	Per cent of total power
B. Power (kw.h. per ton run ore):			
Breaking...............	--	--	--
Intermediate crushing..	0.0066	0.82	4.80
Primary grinding.......	.0450	5.60	32.49
Secondary grinding.....	.0425	5.30	30.69
Classifying and conveying	.0042	0.52	3.02
Flotation....	.0091	1.13	6.59
Concentrates handling....	.0039	0.48	2.80
Tailings disposal........	.0066	0.82	4.76
Water supply.............	.0081	1.00	5.82
Assaying and sampling....	.0006	0.08	0.46
Miscellaneous............	.0040	0.49	2.87
Mill total	0.1306	16.24	94.30
Line and transformer losses	.0079	0.98	5.70
Grand total	0.1385	17.22	100.00

	Dollars per ton ore	Pounds per ton ore
C. Reagents (pounds per ton mine run ore):		
Lime........................	0.0369	4.722
Pine oil....................	.0185	0.237
Xanthate....................	.0076	0.049
Experimental..	.0010	0.003
Total....	0.0640	5.011
D. Miscellaneous supplies...........	0.1602	
E. Miscellaneous	0.0132	
Grand total direct cost, per dry ton of ore milled......	$0.5130	



Water balance for an average milling day during the last six months of 1929

No.	Product	Tons Solids	% Solids	Tons Pulp	No.	Water Additions	
						Tons	Gals/Min
1.	Mill Feed	2500.00	96.8	2582.90	A	632.58	105.43
2.	Primary Mill Disch	2500.00	77.7	3217.50	B.	922.64	153.72
	Bowl Classifier				C.	3044.32	605.04
3.	Bowl Overflow	779.60	14.0	5564.45			
4.	Callow Feed	944.00	12.2	7744.92			
5.	Callow Middlings	165.88	7.6	2180.47	D.	492.70	82.00
6.	Callow Conct	98.21	10.3	953.50	E	228.38	38.00
7.	Callow Tailings	681.39	12.8	5331.96			
8.	Bowl Sands	1780.40	77.6	2817.01			
9.	Circ. Class Disch	2867.39	80.2	3575.50	F	500.41	83.28
10.	Sec Mill Disch	4587.79	73.0	6282.78	G	4850.03	807.17
11.	MacIntosh Feed	3031.18	30.4	9933.50			
12.	MacIntosh Mid'l'gs	310.79	13.1	2372.44			
13.	MacIntosh Tailings	1538.84	22.8	6751.06			
14.	MacIntosh Conct	101.16	19.0	553.47	H	137.08	22.81
	Dexter Tables				I	873.62	145.50
15.	MacIn Coarse Conct	80.81	10.2	786.37			
16.	Gravity Bin Ship's	80.21	90.6	88.53	J.	697.04	116.14
17.	MacIn Fine Conct	100.95	9.7	1040.72			
18.	Dorr Tank Disch	199.16	64.6	308.30	K	1685.02	280.50
19.	Filter Disch	199.16	83.8	237.70	L.	*70.52	11.74

*Water removed

Sources of Water Supply	Tons Water Per Day	Gallons Per Min.	Gallons Per Ton	Gallons Per Day
Mine Ore	83.98	14.00	8.05	20,100
Huacal Dam	5464.16	909.36	523.80	1,309,500
Tailing Dam	4445.18	739.78	426.12	1,065,900
Return Water	2454.28	408.46	235.31	589,200
Totals	12,447.60	2071.60	1193.28	2,983,100

Figure 6.- Diagrammatic flow sheet for separate sand-slime treatment